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DECEMBER, 1951

LITHIUM

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FEATURES

ARTICLES

TRANSACTIONS

The cover this month is a line drawing of crystals of lithium minerals, one is the commercial mineral spodumene and the other a gem mineral tourmaline. The chemical formulas are approximate.

CASE HISTORY OF A SUCCESSFUL TUNNEL JOB

FROM THE EIMCO FILE T1005A

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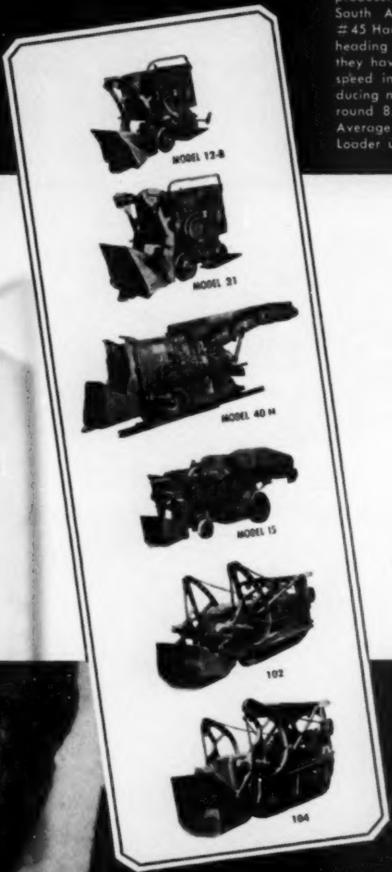

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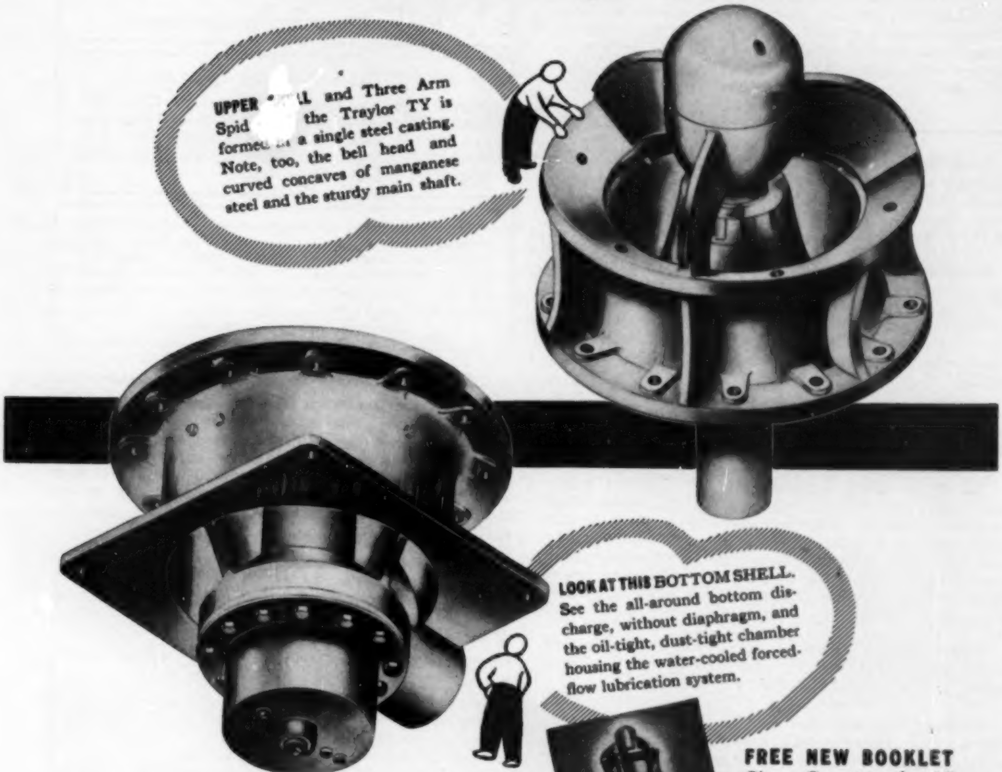

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POSITIONS OPEN

Sales Engineer, 25 to 35, mining or metallurgical training and experience, to sell ore dressing equipment. Reasonable traveling. Salary, $4000 to $5000 a year. Territory, eastern seaboard. Headquarters, New York, N. Y. Y6254.

Engineers. (a) Executive, 30 to 50, with experience and record of accomplishment in administration of large mining and metallurgical operations. Salary, $15,000 a year. (b) Executive with minimum of ten years' experience in operation of heavy chemical plants. Salary, $15,000 a year. Location, New Mexico. Y6226.

Mining Engineer, 30 to 35, to investigate and report on metal properties in Canada. Permanent. Salary, $6000 a year plus expenses. Y6214.

Assistant Mine Superintendent, about 40, preferably graduate, experienced in non-metallics, except coal, desirable, but not essential. Experience with mobile equipment such as shuttle cars and joy leaders highly desirable. Experience and ability in directing mine layout essential for the development of a new mine. Salary, $9600 a year. Housing available. Location, California. Y6199.

Junior Mining Engineer with some operating and exploration experience, for clay mining project. Salary open. Location, East. Y6185.

Assistant Geologist for work on open pit mine maps and sections, field supervision of ore drilling and sampling program and ore property examinations. Salary to start, $3600 to $3900 a year. Location, Texas. Y6172.

Mine Foreman experienced in mining, especially setting, cut and fill and top-slicing. Will assist mine superintendent and take over and run the mine if necessary. Should also have considerable experience in timbering. Will spend about half of day on the mine work and the rest of time on engineering. Salary, $4200 to $4500 a year, plus room and board for single man; if married, will receive a furnished house and $75 a month board allowance. Location, Honduras. Y6130.

Assistant Research Engineer, under 30, engineering graduate, with combustion experience, including coal analyses, to plan and evaluate coal tests, write reports covering analyses, market preparation, etc. Salary, $3600 to $4200 a year. Location, Virginia. Y5970.

Mining Engineer with coal production and preparation experience to supervise installation, operation and maintenance of coal preparation plant. Salary, $12,000 a year. Duration two years. Location, Turkey. Y5891.

Mining Engineer, young, who has had some experience with silver property. Must have had some experience in South America or Mexico. Salary to $8000 a year. Location, Mexico. Y6292.

Mining Consultant to estimate cost of sinking shaft and bringing mine into production, for barrelled mica pegmatite operation. Salary open. Location, New York, N. Y. Y6272.

MINING INSTRUCTOR: Recent graduate preferred. Mining school in the East. Assist in mine surveying, mineral dressing and other courses in mining engineering. Services required for 2nd semester, beginning February 1. Salary depends on experience.

Box K-26 MINING ENGINEERING

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Mining Engineer. Employed. Desires position with established concern. Broad experience examination, exploration, development, operation and heavy construction. Competent technical reports and engineering office procedure. Married, 48, good health. M-651.

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MINING GEOLOGIST, 32, MS Degree, four years' experience with large corporation in open-pit and underground geology, examinations, exploration and development programs. Married.

Box J-21 MINING ENGINEERING

POSITION WANTED: Metallurgical engineer, 9 years' experience in mineral dressing, including 5 years in research, desires responsible position in research or development.

Box K-24 MINING ENGINEERING

FOR SALE

K & E Mining Transit with auxiliary telescope, prismatic eyepiece and tripod. Used only two months.

Box K-25 MINING ENGINEERING

MEET THE AUTHORS

JOSEPH KWONG

W. F. MULLEN

EDGAR L. PIRET

WILBUR T. STUART

J. N. S. Kwong *(Basic Laboratory Studies in the Unit Operation of Crushing, co-author with Messrs. Axelson, Adams, Johnson and Piret, P. 1061)* was born in Canton, China. He attended Stamford University and University of Minnesota. He worked with Minnesota Mining & Mfg. Co. as chemical engineer, with Shell Development Co., and at present is again with Minnesota Mining & Mfg. Co. Chemical processes and plants are of special interest to Mr. Kwong. He holds membership in American Chemical Society and American Institute of Chemical Engineers. Photography and golf are his favorite hobbies.

W. F. Mullen *(Operational Studies in the Pennsylvania Slate Industry, co-author with C. W. Stickler, P. 1097)* was born in Dresden, Ohio and attended Rutgers and Pennsylvania State College. He was a laboratory foreman with E. I. DuPont de Nemours from 1936 to 1942 and was a senior project coordinator with Curtiss-Wright Corp. from 1942 to 1945. From 1945 he has been connected with Pennsylvania State College. Products promotion, utilization of mineral wastes and operation analyses are of particular interest to Mr. Mullen. His favorite hobbies are gardening, amateur astronomy and studying law through correspondence school. An AIME member, he also holds membership in American Ceramic Society, Pennsylvania Ceramics Ass'n., and Illuminating Engineering Society.

E. L. Piret *(Basic Laboratory Studies in the Unit Operation of Crushing, co-author with Axelson, Adams, Johnson, and Kwong, P. 1061)* was born in Winnipeg, Canada and attended the University of Minnesota. He was professor of chemical engineering at the University of Minnesota. He was in charge of Chemical Engineering Dept. of Minnesota Mining & Mfg. Co. from 1942 to 1945. He was a Fulbright professor at University of Paris and Nancy from 1950 to 1951. Mr. Piret has a Chemical Engineering degree from University of Minnesota, Docteur de Universite de Lyon, France and Ph.D. from University of Minnesota. He is an active member of American Institute of Chemical Engineers and a member of American Chemical Society.

W. T. Stuart *(Mine-Drainage Studies in the Iron Ranges of Northern Michigan, P. 1101)* was born in Pueblo, Colo. and attended the University of Colorado. He received a Bachelor of Science and Civil Engineering degree. He has worked with

U. S. Geological Survey in Arizona on ground and surface water hydrology, on ground water problems of alcohol and rubber industries in Louisville, Ky., and on ground water problems of mining industries in Michigan. Water problems of mining industry are of special interest to Mr. Stuart. He holds membership with American Society of Civil Engineers.

J. F. Johnson *(Basic Laboratory Studies in the Unit Operation of Crushing, co-author with Kwong, Piret, Adams and Axelson, P. 1061)* was born in Minneapolis, Minn. and attended the University of Minnesota. He received his Ph.D. in 1948 and was graduated with honors from University of Minnesota. He worked with Allegany Ballistics Laboratory in Cumberland, Mo. from 1944-45 and did graduate work with University of Minnesota from 1945-48. He is at present time with Standard Oil Development Co. Golf and handball are his favorite hobbies. Mr. Johnson is a member of American Chemical Society.

P. E. Landolt *(Lithium, P. 1045)* was born in Brooklyn, N. Y. and attended the School of Mines, Columbia University where he received a Chemical Engineering degree. Besides being a member of AIME, Mr. Landolt holds membership in American Chemical Society, Electrochemical Society, American Association for the Advancement of Science, American Society for Metals, Association of Consulting Chemists and Chemical Engineers. From 1937 to 1938 he was chief engineer with C. S. Sale & Co., industrial engineers. He was responsible for technical investigations and studies in connection with a diversification of industrial problems but with extended investigations on processes for electroforming of metals, including screens; also including associations in operations for the fabrication of stainless steel products. From 1939 to 1940 he maintained his own plant laboratory. From 1940 to 1946 he was president of Lithaloys Corp. and from 1946 to date he is executive vice president of Lithium Corp. of America.

J. L. Stuckey *(Industrial Minerals of North Carolina, P. 1093)* attended the University of North Carolina and Cornell University. He has worked as an instructor at the University of North Carolina in 1921 and from 1922 to 1924 taught at Cornell University. He was a professor of geology at North Carolina State College and state geologist with North Carolina Dept. of Conservation and Development. He is at present director of Minerals Research Laboratory in Asheville, N. C. Besides holding membership in AIME, Mr. Stuckey is member of American Ceramic Society of Economic Geologists and Geological Society of America. Mr. Stuckey has presented other technical papers before the AIME.


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Back-Ripper

These back-rippers which mount on the back of bulldozer moldboards and rip only when the tractor backs up are now available for angling bulldozers, according to an announcement by *Preco Inc.* The four back-ripper housings are welded to the under side of the C frame on angling blade bulldozers, permitting use irrespective of the angle of the blade.

The teeth float on top of the ground when the tractor moves forward, and automatically dig and rip when the tractor backs up. Thus, the dead head back-up time becomes a profitable operation and enables the blade to make a full load on each forward trip. When desired, the teeth can be locked out of the way. **Circle No. 1**

Air Compressor

Heavy construction contractors, and quarry operators, now have available for use the new standard model 365 air compressor, manufactured by the *Jaeger Machine Co.* and featuring diesel power by Cummins. The new compressor, rated at 365 cu ft of air per min at 100 lb per sq in. pressure, utilizes a 165 hp model HRBI-600 Cummins diesel, made by Cummins Engine Co. for its power plant. Engine operating speed of the diesel compressor unit is 1240 rpm. Both the Cummins diesel and Jaeger compressor are mounted on structurally welded main frames. The heavy duty wagon has an auto-steer front axle and can be hauled safely at 35 mph over rough roads. **Circle No. 2**

Core Drill

The new shot core drill designated as the Model KR is a modern engineered core drill designed to recover cores from any formation—hard or soft up to 20 in. in diameter and depths down to 600 ft. Because shot bits are run to destruction and bit maintenance is negligible, operation is exceptionally economical, particularly in cores 3 in. and over. Operation is simple and no special skill or experience is required of the operator. Drilling is always under the complete control of the operator, who can, when necessary, instantly switch to any one of three speeds. This unit is manufactured by *Acker Drill Co.* **Circle No. 3**

Torque Converter

Development of a three-stage hydraulic torque converter transmission which eliminates forward gear shifting on tough grades has been announced by the *Twin Disc Clutch Co.* The new converter, used with a closely spaced transmission, provides smooth, efficient use of power for the trucks that haul 30 ton loads on grades up to 13 pct, field tests indicate. Hauling cycles have been significantly reduced, with substantial savings on axles, tires, gears and brakes. The five-speed transmission provides a reverse gear, a power takeoff drive for the dump body and a low gear ratio for an unusually bad hole. Fourth speed of the transmission is a direct drive and it is in this ratio that the three-stage torque converter has done its payload upgrade hauling on the Mesabi range, even though grades vary on an average from 8 to 13 pct. **Circle No. 4**

Jumbos

Two jumbos were recently placed in operation by *Rogers Iron Works Co.* at an underground iron mine in Missouri. Each of the jumbos has two hydraulic jib arms with self-leveling operators' platforms, independently driven tracks, mast platform adjustable at any height and electric motor driven. The mast folds

to enable the jumbo to pass through a 15 ft entryway. Other Rogers jumbos are available with stationary masts, pivoted booms, jib arms mounted directly on the crawler chassis and with air or diesel engine drive. They are used for drilling, powder loading, scaling, roof bolting and timbering. All models may be easily disassembled into components for entrance through mine shafts. **Circle No. 5**

Hydraulic Test Stands

Superdraulic Corp. who designed this unit fabricates and assembles all types of test stands employing hydraulic pressure. These stands include pumps, relief valves, oil reservoirs, electric motors and incidental valving and piping to meet specific test purposes. These test stands are built to provide test pressure up to 5000 psi, using large volume pumps and up to 30,000 psi, with special booster equipment. **Circle No. 6**

Hydraulic Surface Grinder

This fine precision surface grinder has centralized controls and is simple in design, combining accuracy and efficiency. The vertical adjustment of the wheelhead is .0001 in. divisions and the fine adjustment for the vertical wheelfeed is .0001 in. The wheelhead spindle runs in plain journal and thrust bearings, and is hardened,

ground and tapered at the front end to receive grinding wheel flange plates. The bearings are diamond bored and the spindles are ground and superfinished to within two micro-inches, root mean square. This unit is distributed by *British Industries Corp.* **Circle No. 7**

Classifier

A new classifier with operating characteristics of both the conventional cyclone and centrifuge is now in production by *Equipment Engineers.* Called the Centriclone classifier, it combines the independent variation of residence time and velocity attributed to a centrifuge with the tremendous shearing force. This combination, according to the manufacturer, gives a new character of sizing by allowing sharper cuts to be made than previously were considered possible in mineral dressing. **Circle No. 8**

Free Literature

(9) HYDRAULIC SYSTEMS: Complete engineering data on synclinal type filters for sump or line installation on all hydraulic and low pressure liquid recirculating systems is contained in booklet recently published by the *Marvel Engineering Co.* All parts of the filter are heavily zinc plated except of course the wire mesh filtering element. Special construction of the synclinal filtering element incorporates about 2½ times more filtering surface than is obtainable from a conventional type filter which filters around the outside circumference only. These filters are applicable not only to hydraulic oil installations but are used for any filtration purpose where the liquid is of a noncorrosive nature.

(10) CRUSHING: This attractive booklet covers crushing in the aluminum industry and is issued by *Pennsylvania Crusher Co.* It describes the crushing operations of bauxite, limestone, coke, carbon and cryolite. This booklet points out how the reversible hammermills and reversible impactors provide new concepts of performance and economy for the aluminum industry. The impactor substitutes anvils for cage bars, crushing is entirely by impact. Material, dropping through the impactor's high, centrally-located feed chute, penetrates deep into the path of oncoming beaters, is struck, driven against massive anvils, rebounds, and is again struck by the rotating beaters, shattering under this high-velocity impact-rebound.

(11) BELT CONVEYOR CARRIER: Among the many standard and special carriers described in the new belt conveyor carrier catalog issued by *Stephenson Adamson Mfg. Co.* is the belt training carrier. The training rolls respond instantaneously to any shift of the belt off dead center and tilt the carrier into an angular position with respect to the direction of belt travel. This quickly returns belt to center and the carrier resumes its normal position. Rollers are carefully finished and placed to avoid scoring and injuring the belt. Rollers of different diameters are available—small diameters for low speed belts, and larger rollers to permit higher belt speeds without excessive bearing speeds.

(12) TRUCKS: A 12-page condensed truck catalog has just been released by the *Howe Scale Co.* Specifications and illustrations are shown for the complete line of Howe two and four-wheel hand trucks, trailer trucks, and lift jack systems. This catalog also shows modern Howe construction. The wood parts are cut to pattern, are interchangeable and are varnished to protect against wear and elements. Frames are rigidly braced and extra construction is used whenever it will add strength and life. Howe wheels and casters are available in a variety of styles and construction to meet the varying requirements of individual users.

(13) AERIAL SURVEYS: This booklet issued by *Abrams Aerial Survey Corp.* is a complete explanation of aerial surveying and is written in simple and non-technical language. It serves as a valuable handbook to those who are interested in the subject. Probably, the most interesting section of the booklet is part which covers topographic maps, page 14, and oblique photography, page 9. This booklet covers various products that are made from aerial photographs and ways in which aerial mapping can save time and money on engineering projects. Not only can aerial surveys be used in almost every situation where ground surveys are used, but in many instances the terrain may be so rugged that the only practical way to map an area is from the air.

(14) MOTOR DRIVEN COMPRESSORS: In these motor-driven compressors, full force feed lubrication is provided for all main and connecting rod bearings, crosshead and crosshead pins, lubricator drive, compressor cylinder and compressor piston rod packing. A high capacity, precision built, positive displacement, gear type pump provides lubrication for the main, connecting rod and crosshead bearings. Foreign particles are removed from the oil as it passes through the extremely fine perforations of the removable brass, metal edge type filter elements. Lubrication of compressor cylinders and compressor piston rod packing is handled by a separate, positive displacement, visible feed, mechanical type lubricator. Literature available from *Clark Bros. Co.*

(15) TUBE FITTINGS: Catalog B-151 describes and illustrates the complete line of leakproof Swagelok tube fittings which are available in brass, aluminum, steel, stainless steel and monel. Complete dimensional information, cross-section drawings, installation recommendations and assembly instructions are included. These fittings come completely assembled, finger tight and are ready for immediate use. It is not necessary to perform any preparatory steps on the tubing itself. There is virtually no constriction of the inner wall so that turbulence is held to a minimum. This catalog is issued by *Crawford Fitting Co.*

(16) LUBRICATION: A new 20-page bulletin on seven Keystone specialized lubricants that cover the majority of plant requirements with a minimum of different types and densities. Information presented includes: lubricant characteristics, benefits, uses, densities, specifications, and container sizes. Products covered are lubricants for ball and roller bearings, plain and ring bearings, open gears, enclosed gears, speed reducers, oils for cleaning, loosening and penetrating. This bulletin is available from *Keystone Lubricating Co.*

Mining Engineering **December**
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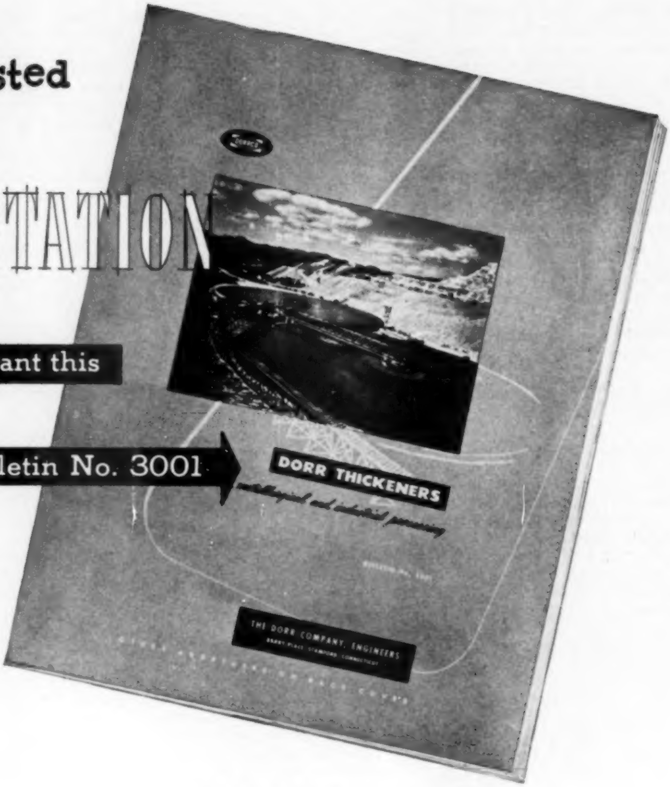

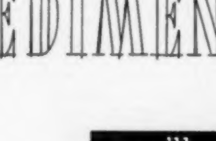

Bulletin #3001, "Dorr Thickeners for chemical, metallurgical and industrial processing" has just come off the press. Containing 28 pages of descriptions, drawings and photographs, it covers the comprehensive line of Dorr Thickeners briefly and factually. If you're concerned with thickening problems and the equipment with which to solve them, you'll find it helpful. Write for your free copy today.

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Trends

WITH the present mobilization program in full swing and the enrollment in engineering colleges dropping off, many solutions are offered to relieve the shortage of trained men in the engineering profession. Industry can absorb about 30,000 engineers annually, according to the Engineers Joint Council, and forecasts indicate that the number graduating will have dropped from 38,000 in 1951 to about 12,000 in 1954. If this condition were to continue there would probably not be enough people with technical training to run industry in four or five years.

In the mining industry there are other factors that aggravate this situation, first, the present trend in mining is a swing to lower grade deposits because the choice ore has been mined; this necessitates better methods and control based on sound engineering principles, thereby raising the manpower requirements. Second, the location and comforts available to men in mining communities are not to be considered the most desirable, therefore, the young men are inclined to spend their money for other types of training offering more satisfactory living conditions.

A feature in the young man's outlook when considering the engineering profession is that a skilled or semi-skilled worker can make more take-home pay after three years training, without any cash outlay on his part, than the engineer can make three years after he graduates from college. The engineer, however, has several thousand dollars tied up in his education.

This could easily be a contributory factor to what H. P. Hammond, Dean Emeritus, Pennsylvania State College, called the "post college slump". This slump is the period when a college graduate takes his first job, and with the feeling that he has no professional status and the lack of guidance by his employer, he neglects his professional education and advancement. The young engineer, upon graduation, works with other men, sometimes younger, that do not have any technical education, yet are in a better financial condition than he. The result is that he loses his drive to learn and advance, becoming a rider in the engineering profession, ultimately a loss to the industry. Others react differently, they leave engineering to become insurance salesmen and real estate agents.

The Engineers' Council for Professional Development plans to spend $20,000 a year for a training program to help the junior engineer to overcome this tendency to slough off in personal development after graduation. This is to be a self-help program in six phases; in-service training, continued college education, community service, professional registration, self appraisal, and selected reading.

The president of Stevens Institute of Technology, Dr. J. H. Davis, recommends that the engineering profession take an active part in interesting students capable of college work in the study of engineering. He also advises industry to be sure that it makes the best possible use of the supply of engineers that is available. Another step would be the development of on-the-job training programs so that people with ability and working on jobs related to engineering can receive training in science.

A positive program for the junior engineer covers two points: an organized educational program, and the development of professional consciousness through the local engineering groups. This would utilize the extension facilities of the universities and colleges and guidance by the older group of the profession.

The industries could provide such programs with the cooperation of nearby institutions with equal advantage to themselves and to their young engineers.

THE Columbia University has announced the launching of a campaign to raise funds for a $22 million Engineering Center. This program was initiated at an announcement dinner at the Waldorf-Astoria Hotel where the principal speaker, Herbert Hoover, addressed some 400 guests.

The Engineering Center will combine teaching, research and practice in engineering and the fundamental sciences, on both undergraduate and graduate levels, to a degree never before achieved.

Both for the purpose of professional training and scientific investigation, the Engineering Center will operate in closest coordination with Columbia's departments of pure science, the social and political sciences, and the humanities.

The Engineering Center will have the "engineer-scientist" as its educational goal. The "engineer-scientist" is defined as a man possessing not only the highest training in engineering technology but also a profound knowledge of the fundamental sciences essential to solution of the increasingly complex problems facing the modern engineer.

In his address, Mr. Hoover said, "But here we meet a great national problem. We do not have enough engineering teachers and we do not have enough students to carry on the nation's work. And we do not have enough research facilities to assure the needed flow of new inventions and improvements."

Artist's conception of the main 14 story building of Columbia's proposed Engineering Center. The Engineering Center will establish a new pattern in technical education, similar to the role of the medical center in training doctors.

THE director of the raw materials division for AEC, Jesse Johnson, in a speech before the AIME in Mexico City, said the agency is now trying to work out a "uranium-is-where-it-ought-to-be" theory.

The job of weaving the vast store of geologic information into a pattern that could be used in relatively unexplored areas to determine the odds for finding uranium was too big for any individual, group or company. The AEC and Government geologists are piecing together information from widely scattered sources and data from a variety of operations.

To further the development of a usable theory for predicting locations of uranium deposits, the Government has set up a program which includes: examining samples from nearly all mines, mills, and smelters to see if any potential uranium producers have been overlooked; evaluation of information uncovered by oil and mining companies in their exploration; loaning technical experts where their specialized knowledge can be used; and giving contracts to universities and private organizations for the development and improvements of metallurgical processes.

Despite the scope of Government activities in uranium exploration and mining, the main job is being done by the mining industry. All the uranium ore produced in the United States is coming from privately-owned mines.

WEST Germany's coal resources lie in the area just north of the valley of the Ruhr River, this area producing 25 pct of the coal of Western Europe. Since much of this coal is suitable for coking, Western Germany has always been a heavy exporter of coal and coke. This export has served to bring Germany raw materials, agricultural products, and finished goods that were not available within her national boundaries.

The Ruhr district has 140 mines now producing 370,000 metric tons of cleaned coal per day. In 1938, the annual production in this area was 127 million metric tons, in 1946 it was 40 million, and estimates for 1951 indicate that 114 million metric tons will be produced.

Miners employed underground in the Ruhr produced 1.97 metric tons per man shift in 1938, and in 1946, 183,186 miners produced 0.987 tons each. The estimate for 1951 shows that 283,000 miners will produce 1.47 tons per man shift.

The increase in production since 1946 has yet to reach the 1938 level, but this lag is due to several factors directly related to World War II. Since 1936, the Ruhr coal mines have been under pressure to produce faster than they could properly open up new coal seams. Normal development work was neglected or reduced to a minimum. During the war, the miners in the 21 to 35 year age group were lost and a large percentage of the housing at the mines was destroyed. At the end of the war, of the 400,000 mine employees needed, less than 150,000 were on hand. The delay in replacing deteriorated mine equipment also has been an important feature in the low production rate.

In the first winter after occupation, 1945 to 1946, the bottleneck was not coal production, but transportation. The war left the transportation facilities completely disorganized. In this same period, the most important item limiting production was the insufficient food supply, malnutrition of the miners being widespread.

The German mining industry has under Allied control come to a position where the bulk of its rehabilitation from war-time damages and malpractices has been accomplished. A large housing program of over 92,000 dwelling units for miners is being planned and fulfillment of this housing program will help return the Ruhr to its former position as an industrial community vital to the economic welfare of Germany and Western Europe.

The winter of 1949 to 1950 was the first one in postwar Germany in which demand for both domestic consumption and export was more than met by production.

SINCE the beginning of DMA's financial assistance program in the search for strategic mineral reserves, a total of 160 contracts has been approved and signed.

The aggregate cost of the contracts approved is $9 million, of which the Government's contribution is $5.4 million, while industry's investment is $3.6 million. These 160 projects are in 20 states and Alaska. Colorado leads with 28 projects.

The exploration program started in April of this year, and of the 160 contracts approved, 73 were for lead and zinc, 23 for tungsten, 16 for strategic mica, 9 for copper, and the balance for other scarce commodities.

THE Revenue Bill was passed by both chambers and signed by the President to become Public Law No. 183, First Session, 82nd Congress.

The new law provides an increase of normal corporate taxes from 25 pct to 30 pct, retroactive to April 1, 1951. On all income in excess of $25,000 the combined rate will be 52 pct with corporate tax ceiling stipulating that for 1952 and later years no more than 18 pct of a company's excess profits income can be taken in excess profits taxes. The excess profits tax base period income credit is cut back to 84 pct for all of 1951 income and to 83 pct for 1952 and later years. The rate for capital gains has been increased from 25 pct to 26 pct, effective January 1, 1952.

The important mining amendments in the law are the expensing of development and exploration costs, increase of coal depletion rate to 10 pct, granting of percentage depletion to numerous nonmetallics, and coal royalties under the same classification as capital gains. The new excess profits tax law places potash, sulphur, and chemical and metallurgical grade limestone mines in with coal and metal mines for exclusion from the excess profits tax of one-half of the unit net income on all production in excess of average base period production. For a mine that was not in production, or was operated at a loss during the base period, one-third of its net income would be excluded from the excess profits tax. Bauxite is included as a strategic mineral exempt from the excess profits tax.

The summary of the provisions of the Revenue Act of 1951 said in part: "Section 309 of the bill provides that the taxpayer, with respect to expenditures paid or incurred after Dec. 31, 1950, in the development of a mine or other natural deposit may elect either to deduct development expenditures, whether incurred before or after the production stage has been reached, in the year when they are incurred, or to treat development expenditures incurred before the production stage has been reached as deferred expenses, to be deducted ratably as the ore or mineral is sold. Such a choice may be made each year, but must be for the amount of net development expenditure made in that year with respect to the mine."

In the expensing of exploration costs, the bill provides that expenditures made to ascertain the existence, location, extent, or quality of any ore deposit prior to the development stage of a mine, the taxpayer may deduct in any taxable year any amount up to $75,000 paid or incurred in that year. The taxpayer may defer any amount up to $75,000 and deduct it ratably as the minerals discovered as the result of the exploration are sold. Expenditures for any four years may be handled either of these ways, but after four years are claimed, additional exploration expenditures must be capitalized as under present law. The amendment made by this section of the bill applies to taxable years ending after Dec. 30, 1950.

A $5000 contract to explore for asbestos in Marinette County, Wis., has been entered into between the Government and the Star Mining Co. of Madison, Wis. The Government will contribute 90 pct or $4500 to the cost of the project. The United States presently produces about 8 pct of its asbestos requirements.

"Sulphur stockpiles are at a minimum for national security and must not be further reduced," was the warning given by L. M. Williams, Jr., president of Freeport Sulphur Co. He stated that although the long-range outlook is favorable, current sulphur production cannot meet the demand and as a result the United States faces a crisis in the coming months.

Copper is in a relatively shorter supply than ever before. With the United States spearheading the rearmament drive, the shortage is being felt here more than in other countries.

Cleveland-Cliffs Iron Co. has asked its preferred stockholders for approval to borrow $15 million to finance part of the company's $45 million expansion program for the three years, 1951 through 1953.

The NPA has requested the release of lead from the Government stockpile, was the announcement made at a meeting of the Lead Consumers Advisory Committee and the NPA. The recommendation was made so that sufficient lead could be withdrawn from the stockpile to make up for the loss in imports and from recent work stoppages.

Sulphur users were ordered by the NPA to limit their sulphur inventories to a 25-day supply, or "a practical working minimum." The action was taken to spread tight sulphur stocks fairly among consumers.

The biggest loan since World War II has been approved by the RFC, for the development of a vast new supply of copper in northern Michigan. The loan was made to the White Pine Copper Co., a subsidiary of the Copper Range Co. The $57 million dollar loan will finance mining of low-grade copper ore deposits at the White Pine Mine.

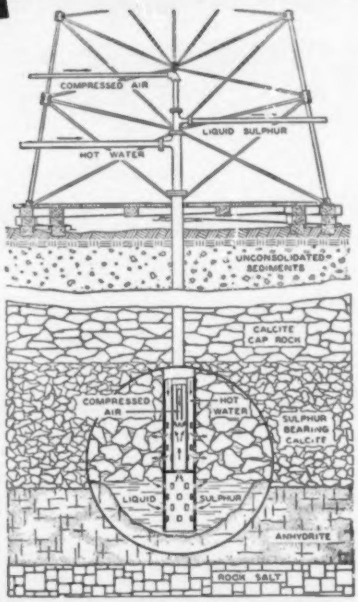
Two ore recovery plants were announced last week: National Lead Co. will process 50 tons of concentrate per day at Fredericktown, Mo., to get cobalt, copper and nickel. The Brown Co. of Berlin, N. H., is buying 27,000 tons of waste copper ore to extract some 9000 tons of sulphur from it.

The Kaiser Aluminum Co. is planning a new $50 million aluminum plant at Newark, Ohio. The plant would have an annual production of 80,000 to 100,000 tons. This plant, if built, would bring the industry's capacity to about 1.5 million tons.

The NPA plans to alter the M-76 authorization to cover the allocation of not only domestic primary lead pig, but also include lead imports and the distribution of lead scrap. It is doubtful that the agency will try to allocate scrap directly to the users in the same manner as primary pig.

SULPHUR

*Interesting Facts Concerning This Basic Raw Material from the Gulf Coast Region



*WELL PIPING

The well equipment consists of pipes of various sizes, placed one within the other and extending from the surface into the sulphur deposit. A 10" or an 8" casing extends to and rests on the top of the cap rock. A 6" pipe, inside the casing, passes below it and reaches into the barren anhydrite. It is perforated at two different levels, separated by an annular collar. The upper set of perforations permits the hot water to enter the sulphur formation and the lower set permits the entrance of the molten sulphur to the discharge pipe fitted inside the 6" pipe.

When a well is "steamed" the hot water passes down the annular space inside the 6" pipe and outside the sulphur pipe and flows through the upper set of perforations into the porous formation. The entire mass through which the hot water circulates is raised to a temperature above the melting point of sulphur. The liquid sulphur being heavier than water, makes its way downward to form a pool and displaces water around the foot of the well, and rises in the well column through the lower perforations into a 3" pipe which is the sulphur discharge pipe. Compressed air released at the bottom of still another pipe fitted inside the 3" pipe rises and mixes with the sulphur column, forming an air lift which raises the liquid sulphur free of water to the surface.

Loading operations at our Newgulf, Texas mine

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Mines: Newgulf and Moss Bluff, Texas

MINING ENGINEERING

EDITORIAL

SWIMMERS NEEDED, NOT FLOATERS

SINCE dad first took us to the ocean we have always seen a plump elderly gentleman who, with supine composure, floats over the crests spouting whale-like to the wonderment of small boys. Floating requires certain physical and mental characteristics that it takes many years to acquire. These are, in order of importance, confidence, beef, and complete relaxation.

Possibly a rough parallel can be drawn between the mining industry and the dignified paunchy gentleman riding the billows. Mellowing in a downtown club he reminisces over his rough-and-tumble youth; when frontiers were being pushed to continental limits and the great mines were discovered on a first come first serve basis. Those were the days of the empire builders. They created the methods, made their own laws, and gambled their very lives on the outcome. For these efforts and sacrifices they shared in the great wealth which was made. There were no comparisons between the small and big operators because they all started small and those with the most financial know-how slowly merged into the big companies of the present.

Today small properties open and close with the fluctuation of metal prices. The big fellow floats on the crest of metal prices with composure and relaxes in the trough. Nothing has been substituted for the vigour and pioneering spirit which wrested the easy-to-find orebodies from the wilderness. Today's hazards are taxes, labor and material shortages, and above all—lower grade orebodies. Industry is reluctant to gamble on even the good geologic bets. The incentive to go out on a limb to bring in a new property is not there, for the individual no longer shares in the profit although the penalty of failure is still just as severe. Higher metal prices are considered the universal panacea.

This would be a pretty good thesis were it not for a lean strong swimmer who provides more to admire than the floater. He is represented by the petroleum industry. Thriving on competition, spending large percentages of operating capital on exploration from which only a few pan out; spending for research; spending for the best technical know-how this nation can produce. The margin of profit in the petroleum industry is small because competition keeps it that way and an operator cannot risk falling behind on methods or miscalculations which might narrow the slender margin between profit and loss. The petroleum industry is not restricted to giant integrated firms for, in the words of W. Chalmers Burns, president of the Hartol Petroleum Corp.: "Through the operations of thousands of independents as against the scarce hundreds of integrated companies competition is guaranteed, and through competition among both large and small companies comes fair pricing for products and their continued improvement. The independent oil company is usually small, flexible in operation, and independent in thought as well as action."

This statement smacks of the mining industry in its heyday. A renaissance of the old spirit with modern variations is needed. For rugged individualism substitute tact and diplomacy, for high grade ore substitute research, for the skilled miners and cheap labor substitute training, mechanization, supervision, and engineering. But there is one thing for which we cannot substitute; that is initiative, the will to fight, to accomplish. For this the industry needs swimmers not floaters.

Gold bearing froth on one of the forty No. 24 Denver "Sub-A" Flotation Cells at MacLeod-Cockshutt Gold Mines, Ltd., Geraldton, Ontario, Canada.

OVER 90% of all FLOTATION PLANTS IN CANADA USE DENVER "SUB-A"

Here's Another Reason Why...

93% RECOVERY OF GOLD IS BY DENVER "SUB-A" FLOTATION

PROBLEM: *At MacLeod-Cockshutt Gold Mines, Ltd., Geraldton, Ontario, gold is intimately associated with sulphides—either in miscroscopic form or in solid solution. Extreme fine grinding of sulphides will not liberate gold to permit maximum economic recovery.*

SOLUTION: Denver "Sub-A" Flotation concentrates refractory sulphides. *(About 93% of the gold is contained in 23% by weight of concentrates.)*

Concentration of refractory sulphides permits preferential treatment (by roasting) which would not be economically possible if directed at whole tonnage milled.

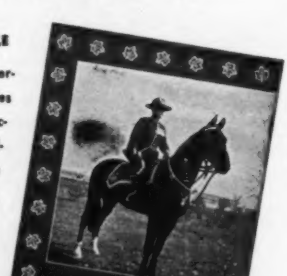
PROOF *of the fact that Denver "Sub-A" Flotation will help your mineral recovery is the thousands and thousands of Denver "Sub-A" installations all over the world and the many operations now changing to Denver "Sub-A" Flotation.*

Denver "Sub-A's" are Standard Flotation machines — flexible to meet flowsheet changes and special problems of your ore.

Write today. Prices and summary of additional advantages will show how you, too, will profit more with Denver "Sub-A" Flotation.

PUBLICATION AVAILABLE

DECO TREFOIL (September-October issue 1951) carries the complete story of MacLeod-Cockshutt gold milling. Your copy will be sent on request. Please indicate your connection with the mining industry.



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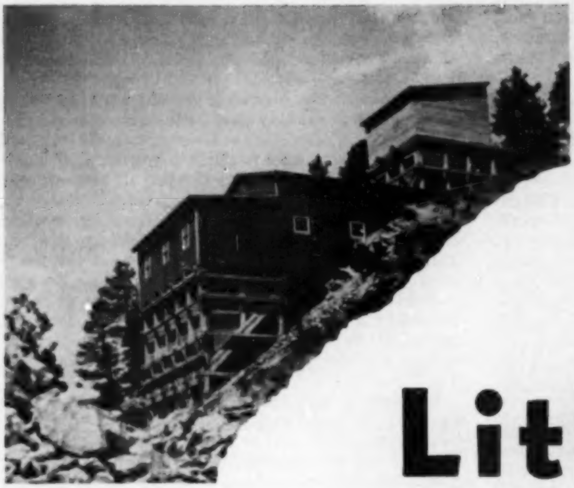


The heavy-media plant of the Lithium Corp. of America at Keystone, S. Dak. The mill treats 15 tons of heads per hour producing a concentrate of 4 to 5 pct Li_2O.

Lithium

Minerals Provide Unique Industrial Raw Material

by P. E. Landolt

MILITARY necessities and economic scarcities, occasioned by the first and second World Wars, led to the search for substitute materials and new products to meet the demands of advancing technology accelerated by research development. Among others, lithium came to attention. Its chemistry had been studied for a long period, but its general use was quite restricted due to limited known sources of raw material.

In World War I, Germany used lithium metallurgically for two principal purposes: 1—hardened lead alloy, B-Metal, for railway bearings as a substitute for lead-tin-antimony alloys; and 2—light strong aluminum alloys, Scleron, in which zinc was largely substituted for copper.

In World War II, due to developments carried on in the United States between the wars, interest in lithium grew rapidly. Major uses were for carbon dioxide absorption (anhydrous lithium hydroxide), hydrogen generation (lithium hydride) for air-sea rescue equipment, Gibson Girl, and lithium chloride in dry batteries. In addition, much research was undertaken to utilize lithium for the improvement of industrial products, for example, ceramic products and in the fields of jet propulsion, organic syntheses, and new alloys such as magnesium-lithium alloys.

Lithium compounds had been used in glass and ceramics and in medicinals for a long time prior to the recent wars. Metallurgical uses, however, were practically unknown except in the literature. Lithium hydroxide had been used for many years in the Edison alkaline storage battery.

MR. LANDOLT is a Consulting Chemical Engineer; Vice President, Lithium Corporation of America, Inc.

This is the fifth in the series of articles on strategic minerals; preceding it are "Cobalt" in January, "Sulphur" in May, "Nickel" in August, and "Tungsten" in October.

The element lithium was discovered by Arfvedsen in Sweden in 1807 while examining specimens of the mineral petalite—a lithium aluminum silicate. This discovery was followed by many scientists, notably, Sir Humphrey Davey, Gmelin, Berzelius, Bunsen, and Matthiesen. A period of nearly a century passed from the date of discovery of the element until its commercial uses became important.

The principal economic lithium minerals are as follows:

1—Spodumene: $Li_2O.Al_2O_3.4SiO_2$ containing approximately 8 pct Li_2O

2—Lepidolite: a lithium mica $(R_2Al(SiO_3)_3)$ containing 3 to 5 pct Li_2O

3—Amblygonite: $LiAlFPO_4$ containing 8 to 10 pct Li_2O

4—Petalite: lithium aluminum silicate containing 2 to 4 pct Li_2O

5—Desert Lake Brines (Lithium sodium phosphate Li_2NaPO_4) as a byproduct of potash and borax recovery

Lithium minerals are found in some quantities on all the continental land masses. From all the data available the importance of these occurrences would be in the following order: 1—North America, 2—Africa, 3—South America, 4—Europe, 5—Australia, 6—Asia.

If dependent solely on the recovery of high grade minerals, a lithium industry of any magnitude could not be developed. So far as is known, such deposits are nonexistent. In earlier years the Etta mine in South Dakota yielded large quantities of spodumene of high quality, but the demand for lithium in earlier years was quite limited. Desert lake brines yield substantial quantities of lithium, but only as a byproduct of major recoveries of potash and borax.

Lithium minerals are usually found in pegmatite dikes which must be of sufficient width to justify mass mining operations. Depth of such mineraliza-

tion increases with the length of the dikes. In most cases the depths vary from 200 to 500 ft; in some, to 1000 ft. The lithium content of these dikes is from 1.0 to 2.0 pct Li_2O.

Up to recent years, lithium minerals have been recovered by hand picking. This method is limited to deposits where the lithium mineral crystal aggregates are big enough to be readily seen and separated.

Many of the most promising sources of lithium minerals are found in inaccessible places; for example, Northwest Territory in Canada. Perhaps in future, when the demand necessitates it, these sources will be developed; but only when the quest for other mineral products opens up these territories to wider and more general exploitation, and with better transport facilities. However, there are adequate deposits now being developed adjacent to good road and rail facilities to provide raw material for the foreseeable demand for lithium products.

The successful flotation of nonmetallic minerals led to trials on lithium minerals, notably spodumene. The U. S. Bureau of Mines cooperated on this with the Black Hills Tin Co., Tinton, S. Dak. During World War II, the Solvay Co. built and operated a flotation mill near King's Mountain, North Carolina. This mill was developed to produce 1000 tons of Li_2O concentrates per month. Actually about 500 tons per month were produced during 1943 to 1945. In October, the Foote Mineral Co. put the former Solvay plant into operation using different methods which are expected to make the operation economical. Flotation is being used to make a separation.

Since the war, application was made of the heavy-media method using a magnetic reagent. Lithium Corp. of America has been operating a mill at Keystone, S. Dak., for about two years, treating 15 tons of heads per hour, and producing a concentrate of 4 to 5.5 pct Li_2O.

This method utilizes coarse crushed material (−1½ in.) subject to the size of the lithium crystal aggregates. It has possibilities for more finely crushed materials. Where mica and feldspar occur with spodumene, difficulties are encountered in making efficient separation of the lithium minerals.

Flotation methods are effective for concentrating such ores, but require expensive grinding, down to minus 40 or 60 mesh. The fine grind, however, tends to slime the mineral, causing a loss of the lithium.

A flotation mill treating 150 to 200 tons of heads per 24 hr is about the minimum for an economical operation. In a mill of this capacity, the average recovery will be 75 to 80 pct, the product containing 5 to 6 pct Li_2O.

In 1946, it was believed that such concentrates could be produced at $4 per unit Li_2O at the mines. Today these costs will be of the order of $7 per unit. These costs are equivalent to 10¢ and 17½¢ per pound of lithium carbonate produced from these concentrates.

Obviously lithium as an alkali element cannot be produced at costs comparable to sodium or potassium. The recoverable raw materials are not as plentiful, nor are they found in nature in concentrated form. Mass handling of materials in large quantities to obtain small amounts of end-product places its burden of cost on such materials.

It is more than likely that many more sources of lithium minerals will be opened up by exploration which will follow the growing interest in and development of lithium products.

Surface exposure of the Cat Lake, Man. spodumene deposit. Diamond drilling has proved deposit to a depth of 200 ft.

The probable demand for lithium products and the mineral deposits readily accessible to provide the raw material is important at this time.

The present and potential demand is approximated in the following table:

Period	Annual Demand for Lithium Products (as Equiv. Li_2CO_3) U.S.A.	Raw Material as Concentrates (5 Pct Li_2O)	R. M. as Mined Material
	Pounds	Tons	Tons
Prior to World War I	400,000 750,000	2,000 4,000	10,000 20,000
World War II	2,000,000 3,000,000	10,000 20,000	50,000 100,000
Post War 1946-1950 (aver.)	1,100,000	5,500	27,500
Potential Industrial Demand	20,000,000	100,000	500,000

Corresponding data for the balance of the world would be undependable, but it is believed that the amount of world demand, other than the United States, (which would follow U. S. developments) would be 10 to 25 pct of the demand for the United States.

The determination of potential demand is at best a guess, but data compiled on known applications are higher than the figures just presented. Estimates of potential demand do not include *any* specific or indicated war uses which have been alluded to from time to time, and published in the technical or lay press.

American Potash & Chemical Co. at Searles Lake, California, has produced lithium sodium phosphate (22 pct Li_2O) equivalent to 1,000,000 to 1,500,000 lb lithium carbonate per annum. This production represents the readily recoverable lithium in the brines. Higher recoveries could be made at excessive costs, and could possibly be doubled. The recovery of this byproduct is not without substantial costs. A rela-

Where Lithium Serves Industry

AGRICULTURE	AIR CONDITIONING	ATOMIC ENERGY	BATTERIES	BEVERAGES	BLEACHING	CERAMICS
Tobacco Culture Soil Moisture Retention Fungicides	Moisture Absorption Dehumidification	Proton Production Tritium Production Power Development Atomic Hydrogen	Primary Cells (Dry Batteries) Storage Batteries (Alkaline Type	Flavor and Tonic Values	Production of Solid, Soluble and Stable Bleaching Agents	Porcelain Enamels Ground Coats and Covercoats on Steel and Aluminum for Acid Resistance Improved Bonding Lower Firing Temperature Pottery Glazes Special Glasses ~
Lithium Carbonate	*Lithium Bromide* *Lithium Chloride*	*Lithium Metal* *Lithium Hydride*	*Lithium Chloride* *Lithium Hydroxide*	*Lithium Citrate* *Lithium Carbonate* *Lithium Chloride*	*Lithium Hypochlorite* *Lithium Peroxide*	*Lithium Carbonate, Manganite, Titanate, Silicate Zirconate and Cobaltite* *Lithium Minerals*

CHEMICALS	GAS PURIFICATION	HEAT TRANSFER	HYDROGEN	IRON & STEEL	MILITARY USES	NON-FERROUS METALS
Production of Miscellaneous Lithium Compounds Catalysts	Removal of Trace Impurities in Helium, Argon, Etc. *Lithium Metal* ~ Carbon Dioxide Removal *Anhydrous Lithium Hydroxide*	Stable Low Melting Point Salt Mixtures	Hydrogen Generation Suitable for Air Transport	Nodular Iron Grain Refinement in Steels *LithiumMetal and Alloys* ~ Desulphurization of Steel	Army Navy Air Force Information Restricted	Chrome Bronzes High Conductivity Copper Castings Bronze, Nickel Silver, Monel and Precious Metal Castings Bearing Metals Aluminum Castings *Lithium Metal Cartridges* ~ Magnesium Alloys
Lithium Carbonate		*Lithium Nitrate*	*Lithium Hydride*	*Lithium Carbonate*		*High Purity Lithium Metal*

PETROLEUM	PHARMACEUTICALS	PLASTICS	REFRIGERATION	RESCUE AND SIGNAL WORK	WELDING
Catalysts Sulphur Removal *Lithium Metal* *Lithium Hydride* ~ Lubricants Low Temperature Greases	Reagent to Produce Antihistamines *Lithium Amide* ~ Reagent to Produce Synthetic Vitamins	Stabilizers Catalysts	Stabilization of Liquid Ammonia	Balloon Inflation *Lithium Hydride* ~ Flares	Fluxes for Aluminum and Magnesium
Lithium Hydroxide *Lithium Stearate*	*Lithium Metal*	*Lithium Stearate* *Lithium Lactate* *Lithium Carbonate*	*Lithium Nitrate*	*Lithium Nitrate*	*Lithium Chloride* *Lithium Fluoride*

Outcrop of lithium deposit at Cat Lake, Man. The exploration of the property is being done by the Northern Chemicals, Ltd.

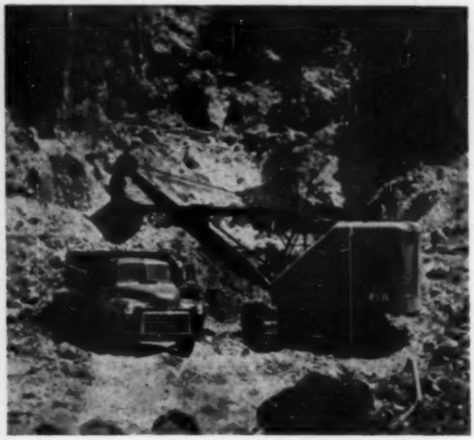
tively elaborate system is needed to make the present recovery.

Any method of recovery must break down the complex silicate in which the lithium is usually found, and may be accomplished by substituting another element for lithium.

The method devised by the Germans used potassium bi-sulphate with considerable success.

Today sulphuric acid has been substituted for the potassium salt with excellent results. By first decrepitating the ore at 1000° to 1100°C (below fusion) spodumene is changed from alpha to beta phase. The decrepitated material is more readily ground than the raw ore. Sulphuric acid is mixed with the decrepitated material which is then heated to approximately 300°C, then leached to produce a 20 pct solution of lithium sulphate, which is purified and treated with soda ash to precipitate pure lithium carbonate. Well over 90 pct of the contained lithium is recovered by this method.

An alternative method is to substitute spodumene for clay or shale in a cement kiln, adding calcium chloride to volatilize the lithium, which is then recovered as an impure lithium chloride. This method was reported by Fraas & Ralston, Bureau of Mines. This method is only feasible where a commercial production of cement can be made simultaneously—at least 1000 barrels of cement per 24 hr. This would produce a very large quantity of lithium salts, but would require an expensive dust and fume recovery installation. Furthermore the purification of the impure chlorides is difficult and expensive. Final costs would not be appreciably different from the acid leach method.



The open-pit mining of Lithium Corp. spodumene deposit at Keystone, S. Dakota. The ore is milled in a heavy-media plant.

The major uses for lithium products may be classified as follows:

1—Ceramics
  a. Glass for X-rays, heat resistance and optical uses.
  b. Porcelain enamels on metals.
  c. Glazes—on ceramic bodies.
  d. Refractories—crucibles.

2—Chemical and allied uses
  a. Organic synthesis (vitamins, antihistamine products).
  b. Lubricants—lithium soaps.
  c. Air conditioning—lithium chloride, lithium bromide.

3—Metallurgical uses
  a. Nonferrous. Copper and copper alloys.
  b. Fluxes for aluminum and magnesium welding.

4—Military uses

5—Atomic energy

Further details on the foregoing outline of lithium uses would better serve to show the role lithium plays in the general applications cited.

Ceramics: Addition of lithium compounds in small quantities to glass frits for porcelain enamels improves the bond between the metal base and the glass or porcelain coating. By adding small amounts of lithium compounds to glass frits for "cover coats," the acid or stain resistance is greatly improved.

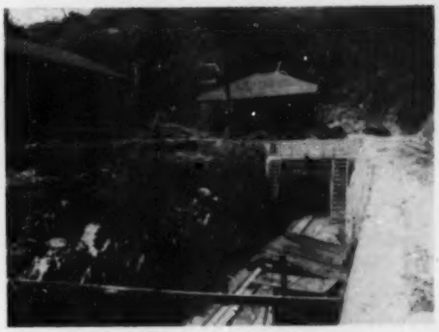
Lubricants: Additions of lithium soaps to lubricating greases widens the range of usefulness of the lubricant. Greases containing lithium have surpassed other types of greases.

Air Conditioning: The lithium halides, particularly the chloride and bromide, have shown marked advantage for moisture absorption. Taking advantage of the thermal properties of these lithium brines, extensive application has been made to air-conditioning units.

Nonferrous Metallurgy: Small amounts of lithium, 0.01 to .05 pct additions, are regularly used in the casting of copper and bronze to eliminate gases and to produce dense castings.

Mineral Status of the Far East

This is the second installment of two-part article on the Far East. The mineral situation in Hong Kong, Indo-China, Thailand, Burma, Malaya, Indonesia, the Philippines, and British Borneo is covered.



The Tiger Portal at the Bawdwin mine, Burma.

by KUNG-PING WANG

Hong Kong (including Kowloon)

Few places in the world comparable in area to Hong Kong (391 square miles) have such a varied geological record. Igneous, sedimentary, and metamorphic rocks are all represented, but igneous rocks, in which most economic minerals have been formed, predominate. The chief minerals are kaolin, argentiferous galena, magnetite, wolframite, cassiterite, molybdenite, garnet, pyrite, mica, hematite, fluorspar, and quartz, of which the first six have been economically mined. Further prospecting is necessary to determine reserves, and the Government has made plans to encourage development of the mineral resources. In view of the Far Eastern crisis, however, the Colony would be doing well if it could maintain its 1949 level of mineral production which normally accounts for about 10 pct of the Colony's foreign trade.

A high-grade magnetite deposit at Ma-an-shan, with reserves of 2 to 10 million metric tons, produced 169,374 tons in 1950. The ore is exported to Japan, while some 40,000 tons of iron and steel products are normally imported every year to supplement domestic scrap. The Colony produced yearly some 5000 tons of lead in concentrates containing 60,000 oz of silver from the Li-ma-hang deposit; resumption of operations depends on funds and equipment. The production of tungsten concentrates has been 200 to 300 tons per year. Tin also has been found, and the Colony has smelted Yunnan concentrates. The Colony is devoid of power and fuel resources and requires annual imports of 600,000 tons of coal and 40,000 tons of petroleum.

Indo-China

Indo-China is mainly an agricultural country, but it appears to have almost all of the essential minerals necessary for industrial advancement. The only minerals and metals that have been sporadically produced are coal, tin, zinc, and phosphorus. The trend toward industrialization was interrupted during World War II, and now a civil war is raging in the northeast part of the country. With the major deposits concentrated in or near this area, the result is low mineral production. Relative stability exists only in the Hanoi-Haiphong delta, the source of anthracite.

Coal resources are 20 billion metric tons, about the same as in India. Proved reserves, predominantly anthracite, are nearer 300 million tons. They have been exploited at an annual peak rate of 2.6 million tons, of which 667,000 tons was consumed domestically while the rest was exported to Japan and South China ports. Present production rate is not much more than 1 million tons a year. Coal is the major power source, although water power should not be overlooked. At least 350,000 kw could be harnessed, mainly in the coal-deficient south at Da Nhim, which is presently being developed. There may be a petroleum field at Qui-Nhon on the east coast, but petroleum, in general, is lacking in Indo-China. The large coal reserves offer possibilities of synthetic oil production.

Indo-China has moderate reserves of high-grade iron ore, approximately 150 million tons, of which one-sixth is proved. The proved tonnages occur mainly in the Tonkin area. The only deposit exploited, although intermittently, is in the Thai Ngugen area. It supplied the bulk of the country's peak production of 136,000 tons in 1939. The second and smaller deposit, used for making native implements, is Phnom Deck, north of Kompong Thom, Cambodia. Known coking-coal resources are small, totaling perhaps 1.5 million tons at one location. In view of a shortage of charcoal, it may be feasible for the country to use electric smelting methods in building a small iron and steel industry. Several thousand tons of metallurgical manganese ore was produced annually before World War II; the output was shipped to Japan. There are a number of alluvial chromium deposits, but the ores must be mined by modern methods. The one at Codein, Annam, is said to have reserves of 2 million tons of 50 pct Cr_2O_3 concentrates. Normal prewar production of tungsten concentrates was 500 tons a year.

Two high-grade bauxite deposits, with combined reserves of 300,000 tons, occur in Tonkin. Copper occurrences are widespread in the north, some with potentialities. Zinc and, to a lesser extent, lead occur north of Tuyenquany, Tonkin. Indo-China has been an important producer of zinc; it has contributed half a million tons of zinc in concentrates since the metal was first mined. Metal production averaged 6000 tons for the 10 year period 1935 to 1945. The only zinc refinery (at Quang Yen), however, has been damaged and must be rehabilitated before production is possible. Likewise, the tin industry has not resumed normal operations; the main smelter produced a peak of 2372 tons of tin in 1938 but was damaged during the war. Antimony was once mined and smelted at the rate of 1500 metric tons annually from deposits in Ta Soi, Annam, and Moncay, Tonkin. In 1943, 20 tons of mercury was recovered from the Tintuc mines alone. Gold production exceeded

300 kg in 1937, surpassing zinc and approaching tin in value for that year.

Of the non metallic minerals, phosphates, mainly apatite, are of major magnitude. Reserves in the Lao-kay area on the Kunming-Haiphong Railway near the Chinese frontier are reported at 100 million tons of 30 to 40 pct P_2O_5 apatite ore. The maximum quantity mined by the Japanese was 100,000 tons in 1943. There were two prewar processing factories for phosphates, one in Haiphong and the other in Mythe; recently the French Government recommended the establishment of a 20,000-ton superphosphate plant in the Danhim Basin. Indo-China had an important glass industry. During the war 100,000 tons of quartz from a mine in Baie de Camranh was shipped to Japan. Large quantities of graphite, some steatite, asbestos, and precious stones have been produced in the country. Indo-China is producing enough cement for its needs from a 400,000-ton plant in Haiphong.

Thailand

Thailand is rich in agricultural resources, but results of a survey by the United States Geological Survey in 1949 indicate that major mineral discoveries are unlikely. Iron-rich laterites and ilmenite sands are widely distributed, yet the largest deposit has reserves of only 700,000 to 800,000 metric tons of 48 to 66 pct iron ore. This ore is used in the country's only iron-smelting plant, a charcoal blast furnace of 25 tons per day capacity recently put into operation. Coking coal is entirely lacking. Of the few small manganese deposits, the best known (on the island of Koh Khram) has but 5500 tons of reserves analyzing 25 pct manganese and 15 pct iron. Thailand cannot expect to establish more than a small iron and steel industry making high-cost products to satisfy a part of its requirements.

The power potentialities are not significant, but plans have been made to utilize the low-grade lignites, the best deposit of which has about 1 million tons of reserves. Oil-shale deposits containing 30 to 330 liters of oil per metric ton occur in the Mae Sot Basin, but their development would be costly. Central Thailand is so level that major water-power sites cannot be developed and potential sites in the north are too inaccessible. The best source of power at present is firewood and, to a lesser extent, agricultural waste. Therefore, Thailand must rely on imports of coal for its needs.

Limestone and building stone are abundant, and the Thai Cement Co., the only cement producer in the kingdom, has two plants equipped to produce 180,000 tons of cement yearly.

However, there are metals that either are being exported or are potentially exportable. Of world importance are tin concentrates, almost all of which are exported. The 1950 output of tin concentrates was about 10,500 metric tons, some 6 pct of the world total. Nearly 800 tons of wolframite concentrates containing 65 pct WO_3 were produced in 1950, and 350 tons of mixed tin-tungsten concentrates were extracted. Combined tin and tungsten production in 1950, although only 60 pct of the prewar peak, is valued at over $30 million. A lead-zinc deposit in Changwat Kanchana Buri, with reserves of some 900,000 tons of ore, is now producing at a rate of 100 tons of concentrates a month analyzing 55 to 70 pct combined lead and zinc in nearly equal proportions. The concentrates are being shipped to Antwerp, Belgium, for smelting. Ilmenite sands are abundant, and some tin tailings contain 60 to 90 pct ilmenite; this product may be exportable, particularly since it contains monazite.

Burma

Burma lacks the resources necessary for a program of industrial development, but many minerals have been exploited mainly for export. The country lacks sufficient land transport facilities to supplement river transport in stimulating mineral production, and unsettled political and labor conditions, including the socialistic tendencies of the Government, discourage capital investments needed. Many of the established mineral enterprises, including Bawdwin and Mawchi, are now idle, and some time will be required for Burma to achieve its prewar production levels.

Petroleum was produced at an annual rate of 260 million imperial gallons of crude during the period 1937-39, mainly from the Yenangyaung and Chauk fields, where reserves are not fully developed. This output is more than adequate, if not exported, to supply the country's oil requirements. The Burma Oil Corp., now partly rehabilitated, produced 15.5 million gallons in 1950 and expects to double this within a year. The coal deposits are small, low-grade and occur in thin seams, but two fields, at Kalewa (Upper Chindwin) and Theindaw-Kawmapyin (Mergui), are said to have combined lignite reserves of 260 million tons possibly suitable for low-temperature distillation and briquetting. Water power also is largely undeveloped, although it is planned to install 100,000 kw of generating capacity.

The country has no substantial reserves of high-grade iron ore; one workable deposit of 3.5 million long tons has been reported. However, iron and steel requirements are small—some 12,000 tons per year—and they can be partly satisfied by processing available scrap.

Several minerals and metals have made Burma famous. The story of the Bawdwin mine is well known, as it is one of the bonanza nonferrous deposits of the world. Original reserves exceeded 10 million tons of 30 to 35 pct lead-zinc ore rich in silver, and while high-grade reserves have been reduced to 3 million tons large quantities of lower-grade ore have not been touched. The magnitude and diversity of production at this property are revealed by the production for 1938: Refined lead, 76,950 long tons; refined silver, 6.2 million oz; zinc concentrates (50 to 60 pct Zn), 61,070 tons; copper matte (42.6 pct Cu), 7800 tons; nickel speiss (31.2 pct Ni and 6.6 pct Co), 3270 tons; and antimonial lead (74.5 pct Pb and 23.1 pct Sb), 1180 tons. Although the property now is idle, a 6-year plan of rehabilitation has been initiated to restore production. There are other lead-zinc deposits that have not been extensively exploited, and new nonferrous bonanzas may yet be uncovered.

Mawchi is one of the largest tin-tungsten mines in the world, and reserves are adequate to produce 2500 tons of contained tin and 1500 tons of contained tungsten per year for at least 10 years. The second largest tin-tungsten producing center is Tavoy. This region has been harrassed intermittently in recent years by guerillas, but in 1938 it produced 2550 tons of tin concentrates and 3057 tons of tungsten concentrates. Burma's total production was about 8000 tons of tin concentrates and 7000 tons of tungsten concentrates in 1938, placing it high among world producers.

The potential contribution of the mineral industry to Burma is indicated by the value of the 1937 mineral production, which was $37 million. In addition to the items noted, the country is famous for gem stones, especially ruby and jade, which are treasured the world over.

Malaya

Malaya does not have enough power and mineral resources to become a highly industrialized country. At present, political instability precludes the possibility of expansion, despite efforts of the government to improve the mineral status. Although the water-power resources are not fully harnessed, they are small, and Malaya's coal and petroleum potentialities apparently are limited. Malaya's known iron-ore resources are the order of 50 million tons of high-grade ore. Although exploited for export, little ore has been consumed domestically. Pig-iron production probably does not exceed 3000 tons per year.

Malaya, however, produces or has potential resources to produce a variety of minerals and metals that are in demand in world markets. The duty paid on tin exports alone constitutes over 15 pct to the government's yearly revenue. Its tin-in-ore and tin metal outputs in 1950 were 57,563 and 68,747 long tons. The 1950 tin metal exports, stimulated by world tension and higher prices, rose to 81,801 long tons; 54.5 pct of the exports went to the United States. Known alluvial tin reserves are at least 1 million tons, while lode reserves have been barely scratched.

During the period 1934 to 1941, Malaya supplied Japan with 1.6 million tons of high-grade iron ore annually obtained from mines in Trengganu and Johore. Production has declined as a result of exhaustion and dislocations resulting from the war, so that the 1951 output probably will be 750,000 to 800,000 tons. From 1935 to 1939 over 30,000 tons of manganese ore was mined for export to Japan, but new reserves must be found for future mining.

Large deposits of high-grade bauxite have been exploited in Johore and Malacca since 1936, and one deposit alone has reserves of 10 million tons; the Japanese were said to have mined 160,000 tons of bauxite in 1943. Ilmenite was first discovered in 1935 and in 1950, 24,915 long tons was produced for export; other rare minerals, however, have been extracted only on a small scale. The Japanese recognized the value of monazite and recovered it during their occupation. Malaya also has valuable deposits of china clay, building stone, basalt, and limestone.

Indonesia

Indonesia is a newly formed republic, comprised of many islands, which are difficult to integrate into a single economic unit and survey for mineral potentialities. Moreover, many deposits that could be worked economically elsewhere have not been exploited because of inaccessibility. Even so, three minerals produced, namely, petroleum, tin, and bauxite, have attracted world attention. The combined value of mineral output in 1950 was close to $263 million, placing Indonesia among the leading mineral producers of the Far East.

Petroleum is the most important mineral and power resource in Indonesia, although virtually all of the production is exported. Petroleum should play a vital role in the industrial development of Indonesia. Three major fields in Sumatra, one in east central Java, and one in southeast Borneo have combined reserves of 1.05 billion bbl. In 1950 crude production was 50.06 million bbl, while refinery production was 53.77 million bbl.

Indonesia's coal potentialities have not been ascertained, but known reserves of three fields, Ombilin mines near Padang (Central Sumatra), Bukit Asem mines near Tangjoeng (S. Sumatra), and the southeast Borneo field are 200 million tons. The only large postwar producer has been Bukit Asem, which produced about 750,000 metric tons in 1950. Little has been done to develop the water-power potential, which is said to be more than 5 million hp.

Indonesia has large reserves of iron ore, a billion tons, mainly on Seboekoe Island in southeastern Borneo and around the lakes of Matanna and Toweti in central Celebes. Most of the ores are medium-grade lateritic. Because of the inaccessibility of the iron ore to foreign smelting centers and the scarcity of coking coal locally, it is unlikely that large-scale iron-ore mining will be undertaken in the near future. There are several manganese deposits, one in west Java, one in central Java near Djocjakarta, and one near Martopera in southeast Borneo. Manganese ore is not now mined, some 40,000 metric tons was produced and exported in 1942. Nickel mines are idle because the better-grade ore has been exhausted, but some 20,000 tons of 3 to 3.5 pct nickel ore was produced annually before the war from the southern Celebes for export to Japan.

Ore bins at Wallah Gorge, 1.6 miles from Namtu smelter, Burma, on Bawdwin mine railway. Stores of electrical equipment were left here by Japanese.

Photo by U.S. Army Signal Corps

A picture of the Namtu, Burma, smelter, the coke plant is on left and Yuille plant on right. Smelter is near Bawdwin mine.

Photo by U.S. Army Signal Corps

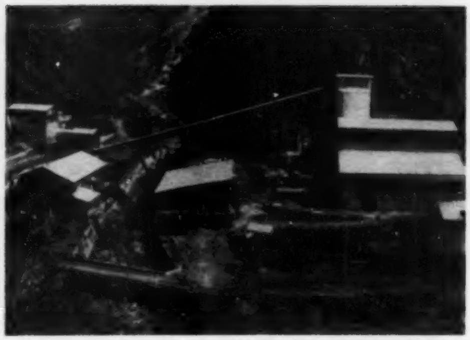
Indonesia is the third largest tin producer in the world, being surpassed only by Malaya and Bolivia. Its 1950 output of 32,131 long tons of tin in concentrates represents about 20 pct of the world total. All the production comes from three islands, Banka, Billiton, and Singkep, named in order of importance. The Banka smelter, which was dismantled during the war, has not been rehabilitated, and concentrates are being exported to Arnhem, Holland, and Texas City, United States, for smelting. Indonesia also has an important bauxite industry, producing about 531,000 metric tons in 1950, mainly from Bintan Island and, to a lesser extent, Kojang Island, where the ore is of better grade. Bauxite, like tin, is mined and milled entirely by mechanized methods; a 55 pct product is now shipped to the United States (75 to 90 pct) and Japan. Potential bauxite reserves exceed 30 million tons. Gold is widely distributed and platinum occurs in the country, but non-ferrous base metals have not been found in large quantities.

A small cement plant in Padang, Sumatra, had a prewar capacity of 240,000 metric tons. The sulphur reserves of 30 to 70 pct ore are about 1 million tons. Phosphate rock also had prewar significance; 34,-085 metric tons of 28 pct P_2O_5 material was produced from a deposit near Cheribon, Java, in 1940. Salt and iodine are extracted from sea water. There are two diamond-bearing districts, one at Landak River in west Borneo and the other in the district of Martapoera in southeastern Borneo.

The Philippines

Although the mineral and power potential of the country have not been carefully surveyed, the Philippines apparently have the economic resources for a moderate program of industrial development. So far, however, mineral exploitation has been confined to gold and certain export commodities such as iron ore, copper, manganese, and chrome, which have little direct bearing on the economy of the country. Postwar rehabilitation and development of the mineral industry have been slow because of the disturbed political situation, inflation and high mining costs, inadequacy of transportation facilities, lack of capital, etc. The Government has initiated a 6-year economic development plan to improve conditions, but it cannot be successful without capital and technical aid.

Coal is found on virtually every large island of the archipelago, but few important deposits have been discovered. Recent estimates place the reserves at only 40 million metric tons, the bulk is non coking. In 1950 the country produced 158,822, metric tons, chiefly from Malangas and Cebu. Petroleum development is still in the potential stage, as initial surveys do not warrant optimism. It is known that the water-power potential and forest resources are relatively large.



The new 550-ton mill of the Lepanto Consolidated Mining Co. on the island of Luzon in the Philippines. The construction of this copper-gold concentrator was started in May, 1947 and was first operated in June, 1948.

The country's iron-ore resources, aggregating a billion tons of medium-grade lateritic ore and 18.6 million tons of high-grade ore, are surpassed only by those of India and China in the Far East. However, metallurgical difficulties must be overcome before the bulk of iron ore can be utilized. The 1950 output of 599,095 metric tons of high-grade ore represents about 60 pct of the average annual production during the period 1935 to 1945. To satisfy the yearly requirements of some 140,000 tons of iron and steel products, the Philippines should eventually draw upon larger quantities of domestic ores to make them. The lack of coking coal may be overcome by making charcoal and electric pig, or sponge iron.

Known chromite reserves are of the order of 10.8 million tons, of which 10.1 million is said to be of refractory grade. The geology of the islands, however, indicates substantial quantities of metallurgical ore. Chromite production in 1950 was 250,511 metric tons of which 208,666 tons was refractory chromite. Manganese ores are also widespread, but known reserves may be only about half a million tons. The

A picture taken before the war of the Demonstration Mill of the Demonstration Gold Mines, Ltd., near Baguio, Philippine Islands.

The first Philippine gold mine to resume operations after the Japanese occupation of the Island, the Benguet-Balatoc Mining Co.'s rebuilt mill represents an outstanding engineering achievement. Badly wrecked during the occupation, the remnants of equipment presented a tremendous challenge to the staff responsible for reconstruction.

yearly production rate in the last few years has been about 30,000 tons, virtually all of which was exported to Japan and the United States.

Gold is the most important mineral produced in the country, and its position was more outstanding in the prewar period. In 1950, 333,991 oz was produced. Production of gold and silver in 1950 represented about 40 pct of the total value of mineral production. The gold industry is experiencing great difficulties because the black-market price has risen to only $50 per oz, while costs are many times prewar costs. Known copper reserves in the Lepanto deposit are over 2 million tons of 4.3 pct, which contains some gold and an indeterminate quantity of low-grade material. Lepanto is now producing at a yearly rate of 15,000 metric tons of copper in concentrates for shipment to the Tacoma smelter in the United States. Lead and silver are being recovered as by-products of gold mining.

The Philippines have sufficient sulphur in nonferrous base metal ores, pyrites and a moderate quantity of guano and phosphate-rock resources. Recently the ECA investigated sulphur resources in the country for possible use by the Maria Christina fertilizer plant and found that production can be greatly increased. Metallurgical and chemical limestone occur widely, and salt is produced from sea water. Construction raw materials are adequate and the cement industry can produce 1.8 million barrels yearly.

British Borneo

This little-known area, comprising North Borneo, Sarawak, Brunei, and Labuan, is extremely backward and inaccessible, making mineral development hazardous. Despite limited exploration, British Borneo seems to have a variety of mineral commodities. The most outstanding is petroleum. The main producer is the Seria field of Brunei and the crude is refined at the Lutong refinery in Sarawak. The country's total refinery capacity is about 40,000 barrels a day, less than half of the crude petroleum production. Although the field had been destroyed by the British and Japanese, the Seria field has been rehabilitated and is producing at six times the 1941 rate. In fact, the country's 1950 output of nearly 31 million barrels of crude is second only to Indonesia. Known petroleum reserves in British Borneo are said to be 500 million barrels. With British access to Iran's oil becoming increasingly dubious, the Seria field is more than ever an important economic and strategic asset for Britain.

Although coal production is negligible, at one time North Borneo, Sarawak, and Brunei, respectively, produced 57,000, 18,000, and 18,000 long tons in a year. The Silimpopon coal field in North Borneo, with reserves of 98 million tons, is the best known. It was worked by the Cowie Harbor Coal Co. between 1905 and 1931 but was later abandoned because of high transportation costs.

Large high-grade manganese deposits, associated with jaspery beds, are widespread in the vicinity of Marudu Bay, North Borneo, with Taritipan as the center. Reserves of more than 1 million tons have been reported, but production had been small and active work ceased as a result of shipping difficulties. Important antimony deposits are said to occur near the surface in western Sarawak. These deposits were exploited on a small scale. However, the Japanese are supposed to have mined several thousand tons of stibnite ore during their occupation. Sarawak also produces gold, but the output has declined substantially from the 28,800 oz extracted in 1934. Occurrences of lead and mercury have been reported in Sarawak.

General view of belt conveyor system for disposal of overburden at Gross-Marble mine of Oliver Iron Mining Co., Marble, Minn. A 10 cu yd walking dragline delivers the material to a rail-mounted screening plant straddling the initial belt conveyor.

OLIVER Hauls Overburden With Conveyor Belts

by J. K. Lovrien

A Link-Belt roller-bearing conveyor system was installed recently at the Gross-Marble mine of the Oliver Iron Mining Co. at Marble, Minn., for removal of overburden and lean ore. The material, consisting of clay, sand, gravel, and glacial boulders, is excavated by an electrically operated walking dragline equipped with a 10 cu yd bucket working from a boom approximately 150 ft long.

This machine loads into a receiving hopper of a rail-mounted screening plant in 45 second cycles. A manganese steel apron feeder under the receiving hopper delivers to an elliptical grizzly, from which the oversize boulders and extraneous material, by way of an apron conveyor, are loaded into refuse haulage trucks.

The undersize from the screen is delivered by a feeder belt to the pit belt of the conveyor system that runs under the screening plant. The initial belt is approximately 1000 ft long and rails along the belt permit the screening plant to be moved as the dragline progresses with the stripping.

The conveyor system consists of seven belt conveyors, one delivering to the other, and a revolving stacker at the end of the 6000-ft belt line, where the material is discharged into a low area. The stacker is a 10-ft gauge, 16-wheel, self-propelled, hydraulic leveling, and revolving machine.

All conveyors are 36-in. wide, conveyor belts are operated at 500 fpm. The capacity is 1140 long tons per hour of overburden containing boulders or lumps of up to 7½ in. size.

There are several transfer points at approximately 90 degrees, from one conveyor to the next. These right-angle turns are necessary to go around the areas to be stripped and removed.

MR. LOVRIEN is Sales Engineer, Link-Belt Co., Minneapolis, Minn.

Each conveyor employs troughing idlers in its carrying run and has rubber-tread impact idlers at all loading points. Each conveyor is built on structural steel frames of standardized design in convenient lengths for easy removal to new positions or transfer to an entirely new location.

The intermediate sections are 21-ft long and may be spiked directly to cross timbers on the ground, to provide a working height of conveyor belt of less than 4 ft above the ground line.

The stacker at the head end of conveyor has a rail-mounted trailer-conveyor section which permits continuous handling of material while the stacker is being moved forward to a new position.

Stripping operations continue day and night and through the winter, as long as weather conditions allow, whereas ore washing plants and actual mining of ore are shut down with the close of navigation on the Great Lakes.

Operations are facilitated by the use of loud speaker system whereby attendants at intermediate points or at the other end of the line, even though a mile away, may quickly learn what new condition has arisen. There is a wire along the entire conveyor line, by means of which the system may be shut down at any point in case of an emergency. Ample lighting is provided to permit operating as efficiently at night as during the day.

Gross-Marble operations are under the supervision of E. A. Friedman, general superintendent, and M. E. Johnson, superintendent.

The overburden conveyor system was engineered and built at the Link-Belt plant in Minneapolis. The self-propelled stacker was built at the company's Pershing Road plant in Chicago.

The dragline bucket delivering stripping into receiving hopper of the self propelled screening plant. Material loaded into hopper is screened and oversize is trucked away.

Junction of initial belt conveyor with second conveyor, showing details of junction house. All belts are 36 in. and rubber tired idlers are used at all loading points.

Below is the 100-ft radius stacker piling overburden in a hollow at the Gross-Marble mine.

Books for Engineers

Detailed Geology of Certain Areas in the Mineral Hill and Warm Springs Mining District. By A. L. Anderson, T. H. Kilsgaard and V. C. Fryklund, Jr. Idaho Bureau of Mines and Geology. University of Idaho. 73 P. Maps, Illus.—Divided into three parts, this pamphlet covers the geology and ore deposits of the Hailey-Bellevue Mineral Belt, the Triumph-Parker mine mineral belt, and the ore deposits of the Mayflower Area. Numerous maps and illustrations are included for each area.

Governme[illegible] of the Copper Industry. [illegible] II. By John W. Douglas. [illegible]—This pamphlet covers in de[illegible] steps relating to cont[illegible] ation of copper and cop[illegible], and is of particular inter[illegible] mary producers, brass mill[illegible] nills, and foundries. Of particul[illegible] current interest is the booklet's coverage of the steps leading to the Controlled Material Plan, into which industry is now once again heading. **Copies can be obtained for $1 from John W. Douglas, Republic Foil & Metal Mills Inc., 55 Triangle St., Danbury, Conn.**

Twenty-Five Years of Building the West. By Arnold M. Ross. Calaveras Cement Co., San Francisco, 1950. 62 P.—This interesting booklet celebrates the 25th anniversary of the Calaveras Cement Co., and was written as a tribute to the firm's founder, William Wallace Mein. Nine chapters cover: the early gold rush days in California, Thomas Mein, organization of the company, formative years, the depression years, cement in World War II, postwar expansion, Calaveras today, and a photographic tour of the whole operation.

Engineering Metallurgy. By A. P. Gwiazdowski. C. C. Nelson Publishing Co., Appleton, Wis., 1950. 247 pp., illus., diagrs., charts, tables, 9¼ x 6 in., cloth, $4.00.—This text is designed to give the student, purchasing agent, production executive, and engineer basic metallurgical information about the nature and characteristics of the commercially important metallic elements and their alloys. Although information on non-ferrous metals is included, the chief attention is given to ferrous metals. The objectives of the book are to present concise and clear definitions and to provide information on the selection of materials and heat treatments.

The Conservation of Ground Water. By H. E. Thomas. McGraw-Hill Book Co., New York, Toronto, London, 1951. 327 pp., maps, tables, 9¼ x 6 in., linen, $5.00—This book is the result of a survey and analysis of the available information on the present development and use of ground water in the United States. It reviews basic hydrologic principles; outlines an inventory of our total water resources; and discusses the experience of over 70 areas (in 35 States) in ground-water development, distinguishing the main types of ground-water problems, and describing the measures which have been or could be taken to improve the situation.

ORDER YOUR BOOKS THROUGH AIME—Address Irene K. Sharp, Book Department. Ten per cent discount given whenever possible. Order Government publications direct from the agency concerned.

Internal Constitution of the Earth. Edited by B. Gutenberg and published by Dover Publications, Inc., 1951., $6.00, 439 pages—Written by eleven of the world's outstanding authorities on geophysics, this is a comprehensive and reliable reference work on the subject. It is a helpful reference work for geologists, geophysicists, mining engineers and metallurgists. Hundreds of invaluable tables, graphs, diagrams, photographs and references are given in this book. It covers such topics as the cooling of the earth and interior temperature; hypotheses on the development of the earth; structure of the crust, and continents, oceans and forces in the earth. This book originally published as Volume VII in the National Research Council's "Physics of the Earth" series, has been completely revised and brought up to date.

Crystalline Rocks of Southwestern California. Issued by Division of Mines, Ferry Bldg., San Francisco, Calif.—This bulletin is accompanied by colored geologic and economic mineral maps. It includes three separate, but technically related reports. The first one is of general character, covering the Corona, Elsinore, and San Luis Rey quadrangles, representing the results of many summer field seasons of mapping by Professor Esper S. Larsen, Jr., of Harvard University. The second report is a detailed geologic treatment of the Cuyamaca Peak quadrangle prepared as a Doctor's thesis at Harvard University by Donald L. Everhart. The third report is on ground water in the bedrock of Western San Diego county prepared by Richard Merriam, a member of the faculty at the University of Southern California.

Ore Deposits of Trepca, Yugoslavia and Its Environments. By Professor Dr. F. Schumacher, $5.00 — The Trepca mine is a development of the Selection Trust, London and is located in the mountains of Southern Serbia. Of Yugoslavia's rich ore deposits, this one is the most interesting. It has one of the most outstanding mineral deposits of the entire world. Both its geological and mineral aspects make this deposit a phenomenon. Geologically it features an eruptive pipe of singular composition. Its violent gaseous explosion filled the blow-out with breccia. The eruptive pipe must be looked upon as a lateral blow-out branching off from the volcanic main neck. Viewed mineralogically, Trepca is one of the world's great repositories. Its crystals are unique both in size and beauty. In the interest of science the author has made a collection of specimens. Favorable exploration led to the establishment of the Trepca Mines Limited by the British in 1927. At the close of World War II, the Trepca was confiscated by the Yugoslavian Government and turned into a state owned and operated plant. Production at Trepca was started in 1930. The mine then rapidly developed into one of the world's foremost suppliers of lead and zinc. Today its excellent equipment make it the focal point of Yugoslavian lead-zinc mining and metallurgy. This book covers such subjects as production and metal content, ore deposits and the general structure of the district.

Engineering Graphics. By J. T. Rule and E. F. Watts. McGraw-Hill Book Co., New York, Toronto, London, 1951. 298 pp., diagrs., charts, tables, 9¼ x 6 in., linen, $3.75.—This text presents the study of engineering drawing in a broader manner than usual. Emphasis is placed on the use of graphical methods for the analysis of engineering problems. Methods of construction of conic sections, roulettes and spirals, graphical scales, empirical and periodic curves, and projective constructions are considered with simple applications to numerical problems. The latter part of the book is devoted to the customary elements of machine drawing which underlie normal drafting room practice.

Thermodynamics: an advanced treatment for chemists and physicists. By E. A. Guggenheim, Professor of Chemistry at Reading University. Published by North Holland Publishing Co., Amsterdam, and Interscience Publishers, Inc., New York in 1950. Available since Sept. 1951. Edited by H. B. G. Casimir, director of the Philips Laboratories, Eindhoven; H. Brinkman, professor at University of Groningen and J. De Boer, professor at University of Amsterdam. This is the second edition.

Application of Geology to Mining at Giant Yellowknife

by J. D. Bateman

At Giant Yellowknife, where high grade gold-bearing orebodies are highly irregular in shape, geology has been applied extensively to the mining of ore. The classical functions of the mine geologist in the fields of exploration and mine development have been extended to guide ore extraction, ensuring "clean" mining and effectively reducing waste dilution.

THE property of Giant Yellowknife Gold Mines Ltd. is situated west of Yellowknife Bay on the north shore of Great Slave Lake, a distance of 600 air miles north of Edmonton, Alberta.

The Giant claims were staked in 1935, the company was formed in 1937, and the main orebody system was disclosed by diamond drilling in 1944 following a geological study of the property by A. S. Dadson,* Consulting Geologist for the company. Production began in 1948 at the rate of 200 tons per day and, during 1950, reached a daily rate of 425 tons. During the first 3 years of operation a total of 366,000 tons was milled with an average grade of 0.79 oz of gold per ton. Work is in progress with an expansion to 700 tons per day in view.

A descriptive account of the geology and gold-bearing shear zone has appeared previously.*

The rock formations in the vicinity of Yellowknife Bay have been subjected to protracted pre-Cambrian tectonic deformation culminating in a series of late faults having a cumulative horizontal displacement exceeding 11 miles. The Giant property is underlain by part of an Archean sequence, several miles thick, consisting of basic volcanic flows and minor intercalated tuffs. The volcanic succession forms the west limb of a major syncline, the flows facing east, but overturned on Giant property to dip west at 65° to 75°.

Orebodies are confined to shear zones up to 200 ft in width, which were formed along early thrust faults. The shear zones assume fold-like attitudes, the larger of which have an amplitude of several hundred feet. The rock formations beyond the limits of the zone of shearing do not reflect the simulated folds, the axes of which are within a few degrees of the strike of the flows.

The schistosity and most of the planar elements in both the shear zones and the orebodies dip west at angles between 65° and 75°, generally corresponding to the dip of the flows. The planar elements within the shear zone system thus dip more or less constantly west whether the shear zone is flat, vertical, or expressed as east or west dipping limbs.

The shear zones reflect the deformation and alteration of the basic volcanic flows into chlorite schists which, in most places, have undergone metasomatic replacement to form chlorite-sericite-carbonate schists or sericite schists. The boundaries between the shear zones and country rock, although often gradational, usually can be defined within a few feet or even inches as they are expressed by the limits of metasomatic alteration.

Orebodies may occupy a small or large proportion of the shear zone and, although they generally conform to the shape of the zone, their morphology is much more complex. Ore boundaries in some instances are sharp and can be delineated with a chalk line; but more generally, a large proportion of the ore boundaries is not visually obvious and can be determined only by the perception acquired by the geological mapping of ore or study of drill cores. Ore shoots in fold-like attitudes may transect the planar elements of the shear zone at any angle; yet the schistosity within the ore shoot may be coincident with that in the enclosing shear zone. Thus it is clear that problems may arise in the delineation of mining boundaries.

Ore generally consists of 20 pct or more quartz with ferruginous carbonates in sericite schist deposited in two dominant stages. The earlier stage consists of quartz with carbonate, pyrite, and very fine-grained arsenopyrite in lenses and bands. The later stage consists of quartz-carbonate lodes, in

* A. S. Dadson and J. D. Bateman: Structural Geology of Canadian Ore Deposits, Can. Inst. Min. Met. Jubilee Volume (1948), pp. 273-283.

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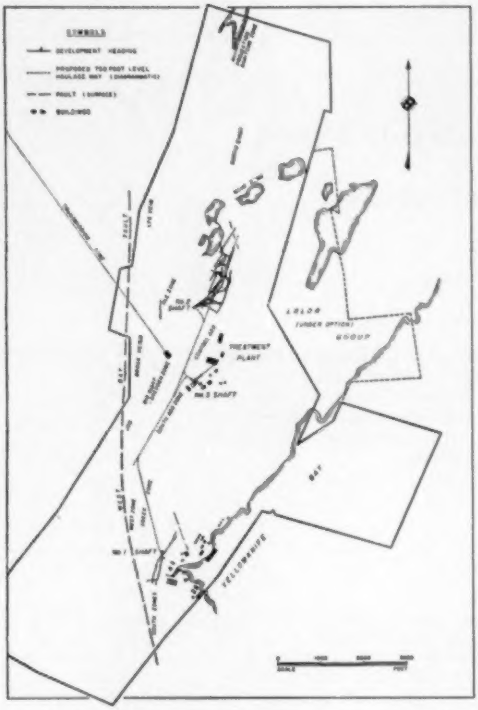

Fig. 1—Plan of Giant property showing relation of shafts and development workings to ore zones.

many cases transecting the earlier stage and distinguished by the presence of sphalerite and the sulphosalt mineral group. Gold is associated with arsenopyrite in the earlier stage, but is more abundant in the later stage of mineralization, in which fine visible gold is sometimes observed. There is a wide variety of ore types, depending upon various proportions of ore and gangue minerals. The sulphide content of an ore shoot may range from 2 to 7 pct. Orebodies are dislocated by postore faults, the largest of which, the West Bay fault, has a horizontal component of movement ranging from 16,000 to more than 25,000 ft.

There is a broad structural control of orebody distribution in the shear zone system with orebodies developed in structural anomalies of the system expressed as steeps or flats, and particularly, where the zone is sufficiently deformed to effect reverse dipping limbs. Statistically, more ore is found within or near crests than troughs within the shear zone system.

There is, further, a control of ore shoots exerted by lithology with ore mineralization showing a preference for some members of the volcanic formations over others. The length, depth, and pitch of individual shoots therefore is controlled generally by the trace of the intersection of the shear zone with the flows. Ore shoots are from 200 to 2000 or more ft in pitch length, ranging in width up to 100 ft.

Exploration

Exploration on the property is conducted by surface diamond drilling, which is planned and executed by the consulting geologist and the chief geologist. The locations and depths of holes are determined by the statistical probabilities of ore shoot distribution after assessing all geological considerations. Although diamond drilling programs are planned well in advance of the commencement of drilling, the location of each drillhole often is determined by the results of the immediately preceding hole. This flexibility is considered essential for efficient use of the diamond drill as an exploratory tool. A total of 130,000 ft of surface diamond drilling has been completed to date, and this figure is being increased by annual projects of 10,000 to 15,000 ft.

Exploratory surface diamond drilling merges with development drilling as ore disclosures are probed further in detail to arrive at preliminary estimates of tonnage and grade and to provide adequate information for the guidance of primary underground development headings.

Development

Lateral development has been undertaken from two shafts 1 mile apart; in addition, an intermediate production shaft, No. 3, was completed in 1951. The relations of the shafts to the ore zones on the property are shown in Fig. 1. Current development plans call for interconnected workings from the three shafts on the bottom or 750-ft level, which will be extended over a distance of 2 miles. Ultimately an additional mile of workings serviced by a fourth shaft will be required for the development of the northern part of the property. Limited probing has shown good ore-bearing conditions to a depth of 1250 ft, and there is no known geological limit imposed on the depth to which ore will be found.

Primary development headings from each shaft are planned by the geological department, and such headings are carried above or below, within or beside ore shoots.

The exact locations of headings with respect to ore shoots are determined by the most suitable approach for detailed underground diamond drill definition of ore, and also by the ultimate utility of the heading for the mining of ore.

As no two ore shoots are similar, the development approach to ore on each level presents special problems involving the interpretation of geological data. In the development of shoots pitching at a low angle, crosscuts commonly are driven at the intersection of the bottom of the shoot with the level, requiring critical projections of geological information. In addition to driving in ore, drifts are carried beneath an ore shoot or above the apex of a shoot, the location of the heading being determined by the projection of diamond drill data. With few exceptions all development headings are driven on predetermined lines; but in practice, diamond drilling may be done at intervals of 100 ft, and the line drive adjusted accordingly as required.

Underground Diamond Drilling

Underground diamond drilling is planned and executed by the geological staff. All development drilling is done on standard section lines at 50-ft intervals with a sufficient number of holes to provide a reasonable outline of the ore shoot. Fig. 2 is a typical development section illustrating cross-sectional diamond drilling of an ore shoot. It will be noted that the drift is simply a small opening in the ore shoot providing access for diamond drilling and, later, for

a stope raise to be driven from the level below. Where the disposition of the ore is more than usually complex, cross-sectional drilling is carried out at 25-ft intervals. Following completion of development diamond drilling of an ore shoot, the ore is reclassified from indicated to developed reserves providing there are accessible openings below. A total of 150,000 ft of underground diamond drilling has been completed to date. Approximately 10 pct of this drilling is for exploratory purposes, 50 pct is classified as development drilling, and about 40 pct is required for detailed preparation of stope layouts.

Mining System Selection

Because of the varied disposition of ore, each shoot presents a specific problem of extraction. The mining system is determined, therefore, not so much by considerations of ground support as by the attitude and size of the ore shoot. All present underground workings are at relatively shallow depths, and there are no mining problems involving support of the superincumbent load. As much as 10,000 sq ft of backs have been exposed in stopes with good support throughout. Some upper level stopes are in permanently frozen ground and there are some technical problems encountered in the permafrost zone, particularly with relation to the extraction of surface pillars.

Shrinkage stoping is indicated wherever the ore can be drawn at angles steeper than the angle of rest for the broken material. In low angle or complex ore shoots, open stoping is carried out providing the height of exposed backs is maintained within reasonable limits. Cut-and-fill stoping is being initiated for the larger ore shoots with low angle attitudes. Individual stopes at No. 2 Shaft, the current source of production, range from 10,000 tons to 150,000 tons of ore, grading from 0.40 oz per ton in gold to more than 1 oz per ton. With further increases in milling rate, lower grade orebodies will be brought into production from other shafts.

Diamond drilling of ore shoots from development headings is accomplished in sufficient detail that contour plans of the ore outlines can be drawn at 10 to 20-ft vertical intervals as required. Contour maps assembling all information pertinent to the

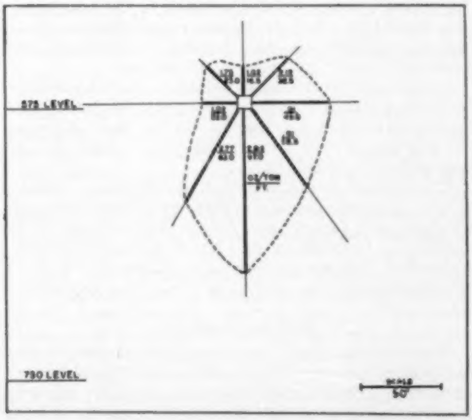

Fig. 2—Cross-section showing diamond drill definition of ore. Giant No. 2 shaft, section 1800N.

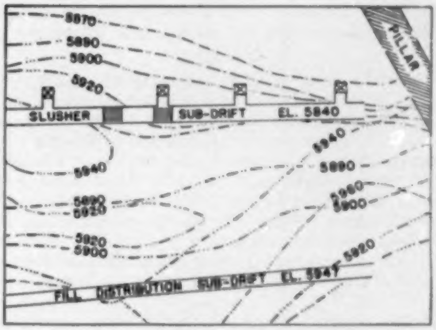

Fig. 3—Plan showing stope development layout superimposed on ore outline contours.

extraction of the ore shoot are prepared by the geological staff and forwarded to the engineering staff for consideration of the mining method. A mining layout is then designed by the engineering department in consultation with the mine superintendent and approved by the mine supervisory staff and the chief geologist. Fig. 3 is a typical stope preparation layout showing the positions of draw points with respect to ore boundary contours.

Mining Boundaries Control

With the commencement of preparation of a stope, geological mapping is started and is carried on throughout the life of the stope. The primary purpose of this mapping is to provide guidance for the mining of ore. All pertinent geological, assay, and diamond drill information is incorporated in 20-scale plans carried by the stope geologists. Test holes are laid out regularly by the geologists and plotted on the plans. Geologists mark with chalk all ore boundaries exposed in the stope each day and indicate the extent to which ore may extend into the walls of the stope as determined from test holes and other data. Fig. 4 is a typical cross-section through four stopes; and in examining this section, which faces north, it should be kept in mind that the lineation both in the ore and within the shear zone dips steeply west (to the left) at 65° to 75° regardless of the dip of the ore. Access to the lower stope, No. 314, is by a sublevel driven above the 425-ft level and from which the ore has been outlined by diamond drilling. The pendant of ore below the sublevel is being removed by shrinkage stoping, the broken ore being drawn on the 425-ft level. Above the sublevel the footwall of the ore is too flat for shrinkage mining, and plans are under way to remove this section up to the 250-ft level by cut-and-fill. The section above the 250-ft level was mined in 204 stope by open stoping as the ore is relatively narrow and of limited height above the level. The irregular section of ore to the east was mined in 218 stope by open stoping and movement of the ore to the draw point with a scraper. In this instance it was essential that the draw point be placed at the lowest part of the ore. The steep ore section in 212 stope was mined by shrinkage stoping and the ore removed by mucking machine crosscuts, one of which is illustrated in the section.

It is clear that no standard procedure can be used in the extraction of ore and that considerable detailed information is required preparatory to an at-

tack on an ore shoot. It is the responsibility of the geological department to provide this information.

Dilution

When a mining plant is operating at capacity or at a fixed rate, each ton of waste that is broken in a stope and ultimately finds its way to the ore pass displaces a ton of ore in the mill and is directly reflected in production figures. The control of dilution is therefore an important factor in the economics of mining. The primary control is exercised by the most efficient mining system for the clean extraction of ore compatible with the cost of mining. The secondary control is exerted through the chalk-line definition of ore boundaries by the stope geologists, see Fig. 5.

In the mining of very irregular ore, a certain amount of dilution is to be expected no matter what mining system is employed; but if the general shape and irregularities of ore are made reasonably clear to the mine supervisory staff and to the miners themselves, inherent dilution can be greatly minimized.

The mine geologist is able to assess the quality of broken material, knowing its source, and is therefore able to distinguish broken ore from broken waste. For this purpose the geological department maintains a blackboard at each ore and waste pass listing all working places; on it broken material arriving in trains is assigned as ore or waste. In the preparation of new stopes, waste work is completed as far as possible before entries are made into ore to avoid mixing ore and waste in draw points.

Sampling

Sampling of development headings, stopes, and of broken material trammed from draw points is carried out under the direction of the geological department. Areal coverage of freshly broken faces by

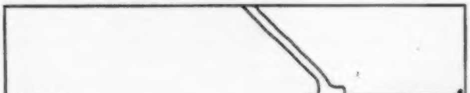

Fig. 4—Vertical cross-section through four stopes. Giant No. 2 shaft, section 850N. Stoped ore, solid black.

Fig. 5—Chalk line definition of ore.

chip sampling gives a more reliable determination of ore grades than cutting channel samples. Current and cumulative sample averages are maintained for each working place. The primary purpose of mine sampling is to determine the grade of material broken and trammed, the broken ore being sampled from cars. The secondary purpose is to assist in the visual selection of mining boundaries. With few exceptions there are no assay walls to orebodies. Sufficient experience has not yet been gained to determine a correction factor for stope sampling. The cumulative car sample average for the first 390,000 tons milled is 0.791 oz in gold per ton as compared with a calculated mill head of 0.794 oz per ton, a very satisfactory check.

Conclusions

The effective use of geology in all stages of the underground mining operation at Giant Yellowknife requires a close liaison between the geological and mine supervisory staffs. Geologists supply the mine staff with stope plans (and sections where necessary) which are compilations of geological data, incorporating significant assay information, classifying ore, and showing test holes. Such plans usually include ore boundary contours for 10 to 20 ft above current working faces, as their main purpose is not so much to record past events in the operation of a stope as to assist in predicting what is about to happen. Meetings are held twice weekly by the geological, engineering, and mining departments at which all current problems are discussed.

The position of the geologist in the control of mining boundaries is essentially advisory, involving responsibility without authority. On the basis of present experience one stope geologist is required for each 300 tons milled daily.

It is difficult to estimate in actual tonnage the degree of clean mining and control of dilution exerted by geological supervision; but it is considered that dilution is reduced by at least 6 pct and that an equivalent tonnage of ore is made available that otherwise would be missed in daily mining operations.

Acknowledgment

In conclusion, the writer would like to express his indebtedness to the late A. K. Muir, General Manager, whose cooperation has made possible the use of geology in the extraction of ore at Giant, and to A. S. Dadson, consulting geologist, many of whose ideas have been incorporated in this paper.

Basic Laboratory Studies In The Unit Operation of Crushing

by J. W. Axelson, J. T. Adams, Jr., J. F. Johnson, J. N. S. Kwong, and E. L. Piret

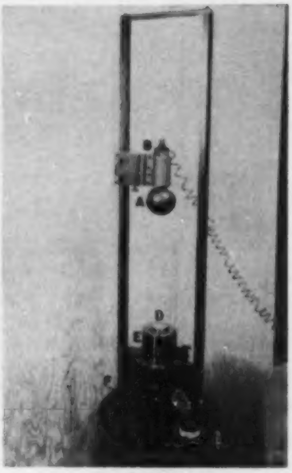

Fig. 1—Drop-weight crusher assembly.

CRUSHING has always been a major operation in the chemical and metallurgical industries, yet little is known about the theory of crushing, and today, the design of crushers is still based almost entirely on empirical knowledge and accumulated practical experience. In view of the increasing national need for the economic working of poorer grades of ores, the lack of a fundamental understanding of this unit operation hardly presents a satisfactory situation.

Basic investigations of crushing have been concerned mainly with three phases of the problem, 1—the mechanism of the fracture process itself, 2—the particle size distribution of the crushed product, and 3—the relationship between the energy input and the amount of new surface produced. Probably adequate information on these phases will be required for a comprehensive understanding of the process of crushing. A bibliography covering these phases will be given as well as a review of the work done in recent years at the University of Minnesota on the relationship between the energy input for crushing and the amount of new surface produced.

Fracture of Solids

This phase of the crushing problem is the most fundamental because it is concerned with the actual mechanism of fracture. The main problems in a study of the fracture process are concerned with the questions of why and how fracture occurs and why there is such a discrepancy between the actual and theoretical energies needed in a fracture process.

In considering the how of fracture, Poncelet[illegible] carried out experiments on the crushing of glass under compression. Actual photographs were taken of the glass plates in various stages of fracture. From these experiments Poncelet was able to postulate a probable mechanism of fracture.

Probably because of the quantitative nature of the problem, considerable effort has been expended to find an explanation for the low tensile strength of materials and the conversely high energy for crushing. Tensile strengths are often only 1/500 of the theoretical while the energy for crushing is usually at least 500 times the theoretical. A review of the work to explain the low strength of glass is given by Weyl,[illegible] in which the work of Griffith,[illegible] Joffé,[illegible] and Powell and Preston [illegible] are discussed. In a recent paper Seitz[illegible] holds that the presence of flaws and fissures, as originally expounded by Griffith, is the most probable reason for the low strength of metals.

Since a mass of data has been accumulated on the fracture strength of materials, several investigators have attempted to correlate this data in the form of an equation. Glathart and Preston,[illegible] Taylor,[illegible] Poncelet,[illegible] Machlin and Norwich,[illegible] and Frederickson and Eyring,[illegible] have all considered this problem, and the final relationship has always been that the stress for fracture is proportional to the logarithm of time. Murgatroyd[illegible] arrived at a very similar relationship by a different method.

The study of the fracture process has used many methods. Among some of these are the thermodynamic approach used by Fürth[illegible] which was based on the work of Born[11] and later used by Saibel,[illegible] and the statistical method applied by Fisher and Hollomon.[10]

In recent years metallurgists have concerned themselves with the problem of brittle fracture as opposed to the usual plastic failure of metals. The great amount of interest shown is reflected by the number of contributors to the recent symposium on the fracture of metals.[1] Repeated failures of welded ships during World War II have accelerated these studies.[illegible]

The shifting of thought from an emphasis on the

J. W. AXELSON, Member AIME, Johns-Manville Corp., Manville, N. J., J. T. ADAMS, JR., Atlantic Refining Co., Philadelphia, Pa., J. F. JOHNSON, Standard Oil Development Co., Linden, N. J., J. N. S. KWONG, Shell Development Co., Emeryville, Calif., were formerly with the Department of Chemical Engineering, University of Minnesota, Minneapolis. E. L. PIRET is with the University of Minnesota.

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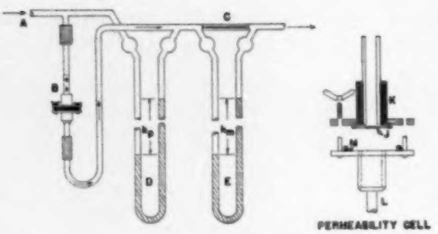

Fig. 2—Air permeability apparatus.

importance and significance of the concept of a critical stress and tensile strength measurements in the study of fracture of materials to an interest in the understanding and measurement of the energy required to produce fracture is believed by the authors to be highly desirable. This will tend to bring the work on fracture and on crushing closer together.

Size Distribution of Crushed Material

The size distribution of crushed materials has been studied for many years[illegible] but the information obtained was generally used as a measure of the surface area or the state of the final product and not as a means of studying the actual mechanism of crushing. Screen analyses run on crushed samples have established a definite relationship between the weight retained on a screen and the size of the screen opening. Schuhmann[44] presents some of the latest work on this. The same relationship has been used by Bond[10] in deriving his method of evaluating surface areas of crushed materials from a screen analysis.

Gaudin and his associates[illegible] have used their data on size distribution to derive some hypotheses on crushing, making use of various postulates by Andreasen[3] and Bennett, Brown, and Crone[5] to synthesize size distribution curves which agree very well with experimental data. Epstein[17] has used an idea similar to that of Bennett, et al, in his recent study of the size distribution of a crushed solid. From the experimental data which they obtained Gaudin and his associates made the amazing conclusions that "1—The size distribution of broken fragments made by a single fracture is such that the new surface on each grade is the same; and 2—Multiple fracturing results in a size distribution such that in the fine sizes, the surface per grade is the same in every grade; in coarse sizes the surface per grade decreases gradually with increasing size." This is about the only work that attempts to make use of the size distribution of a crushed product as a key to the understanding of the fracture process.

Energy Input Vs. New Surface

Prior to 1928 a considerable portion of the literature in the field of crushing[illegible] was concerned with the controversy between the supporters of Rittinger's law and the advocates of Kick's law. Even though Gross and Zimmerley[illegible] seemed to have demonstrated quite clearly in 1928 that Rittinger's law is valid, at least for quartz and for some conditions, the dispute over the two laws still continued.[7,14,illegible]

Energy Measurement: Studies of the crushing process have been complicated by the fact that a valid measurement of the energy input to a crushing operation is difficult to measure. Several attacks are possible. One can try to measure, for example, the energy input to a commercial machine,[illegible] the energy input to a bed of particles in a simple crushing device,[illegible] the energy input to a single particle,[5] or measure the difference in the total energy levels of the material before and after crushing. The last is the most basic but is difficult to perform today.

Surface Area Measurement: Although surface measurements made on fine materials by means of gas adsorption and the electron microscope are probably a close approach to the true value,[4,illegible] these methods were not available until recently. The most common method of determining the surface area of particles has been the indirect one of obtaining a screen analysis and more or less arbitrarily assigning a shape factor to correct for the assumption that the particles were perfect spheres or cubes. The large errors that can arise from this assumption have been demonstrated by Gross and Zimmerley[illegible] and Gaudin and Hukki.[illegible] Bond[10] has developed a graphical method of determining areas from a screen analysis which results in a much improved value for the surface of the material smaller than the smallest sieve size. Martin[illegible] in 1925 and Gross and Zimmerley[illegible] calculated the surface area of crushed quartz particles by measuring their rate of dissolution in hydrofluoric acid. While this method undoubtedly results in good values, it is not easy to perform, cannot be used on heterogeneous material, and the solvent used has to be specific for the material tested. Furthermore, some of the material is dissolved during the area determination and so cannot be used for additional experiments.

Quite a number of investigators[1,4,15,illegible,43] have calculated surface areas of particles by the permeability method first introduced by Carman[illegible] in 1937. Although this method probably does not give a measurement of the true surface area, it is rela-

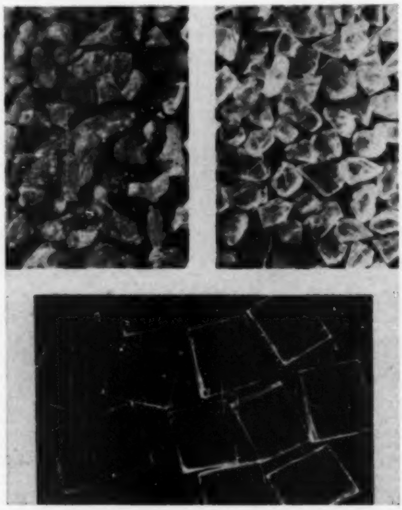

Fig. 3—Micrographs of materials crushed. Top left—14/20 mesh labradorite, X3.5. Top right—8/10 mesh fluorite, X2.3. Bottom—Glass pieces, 0.9x0.9x0.25 cm.

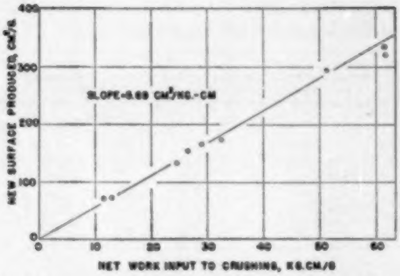

Fig. 4—Energy-new surface relation for impact crushing of milky vein quartz.

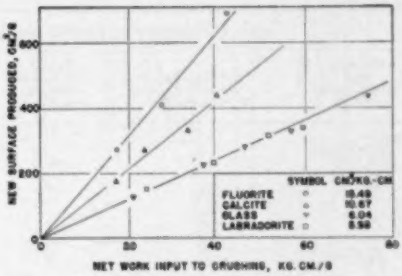

Fig. 5—Energy-new surface relation for impact crushing of several brittle materials.

tively simple and easy to use. Turbidity measurements of surface have been highly developed[illegible] but are not applicable to the larger mesh sizes usually encountered in any crushing experiments.

The most promising and most recent method for surface area determination is the gas adsorption method. Wooten and Brown[illegible] have adopted the general adsorption method developed by Brunauer, Emmett, and Teller[13] to the determination of small areas, and this has considerably increased the range of areas which can be measured. This method was used by Gaudin and Hukki[illegible] and at the University of Minnesota[illegible] in studies on the crushing of quartz.

During the past decade several investigators working under E. L. Piret at the University of Minnesota have conducted a fundamental study of the process of crushing. Most of this work has been concentrated on crystalline quartz, but other minerals have also been investigated. The primary interest has been the relationship between the energy input and the new surface produced in a crushing operation.

Permeability Surface Measurement: A considerable portion of the work done at the University of Minnesota[illegible] made use of a drop-weight crusher, see Fig. 1, similar to that used by Gross and Zimmerley.[illegible] In this apparatus the material is placed in the steel mortar *E*, and is crushed by the impact of the steel ball *A* as it drops on the cylindrical plunger *D*. To prevent rebound of the ball, three aluminum wires are spaced equally under the mortar. When the ball is dropped with no material in the mortar, the deformation of the wires is taken as a measure of the kinetic energy of the falling ball, and it is assumed that a similar wire deformation with material in the mortar represents an equal amount of energy. Therefore, in a crushing experiment, the energy utilized for crushing is the difference between the kinetic energy of the falling ball and the energy represented by the deformation of the aluminum wires. This latter energy is obtained from the calibration curve of wire thickness vs. energy input.

Fig. 2 is a diagrammatic sketch of the air permeability apparatus used to measure surface areas in the initial work. The air enters at *A*, flows through the bed of material *B*, through the flowmeter *C*, and finally to a vacuum pump. The pressure drop, hp, across the bed of material is measured by the water manometer *D*, and the rate of air flow is measured by the pressure drop h_m, across 300 cm of 1 mm ID glass capillary tubing represented by *C*. If these pressure drops and the thickness and porosity of the bed are known, the surface area of the particles can be calculated easily. In some of the measurements on coarse material, a water permeability apparatus of a similar type was used.

Fig. 3 is a photograph of some of the materials crushed. The samples were always in the size ranges shown close to them.

Table I gives a summary of the results of a series of crushing experiments with a variety of minerals using sample weights of 12 to 50 g. Figs. 4 and 5 are plots of the relationship between the new surface produced and the net work input for most of these materials. The slopes of the lines in these plots indicate the amount of new surface formed per unit of energy input, and this value is seen to decrease from fluorite through calcite, glass, labradorite, and quartz. These same values are given in col. 6 of Table I for all the materials, and Fig. 6 is a plot of these values vs. the Mohs' hardness of the materials. Although a few materials do not agree too well, the plot does show that the amount of new surface formed per unit of energy input decreases as Mohs' hardness increases and is essentially a straight line relationship.

All of the materials considered so far have been supposedly brittle materials. The crushing of a nonbrittle but crystalline solid is now considered. When sodium chloride was crushed as described in this section,[1] the relationship between the new surface and the net energy input was found to be a curved line as shown in Fig. 7. Since sodium chloride has considerable plasticity under certain conditions, plastic deformation was suspected as the reason for this unusual behavior. X-ray photographs of the crushed salt showed a definite asterism of the Laue spots which were not present in the uncrushed material or in the crushed brittle solids. This asterism is taken as evidence that plastic deformation did occur. Since no part of the energy causing plastic deformation appears as new surface and the amount

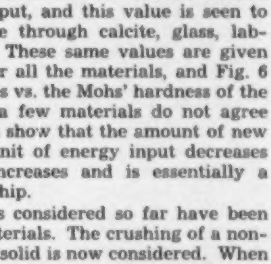

Fig. 6 — Relationship between the Mohs' hardness of various minerals and the surface produced per unit of energy input.

Table I. Summary of Results Using Drop-Weight Crusher and Air Permeability

Material and Experiment No.	Net Work to Crushing Kg-Cm per G	Initial Surface of Sample, Sq Cm per G	Surface of Crushed Product, Sq Cm per G	New Surface Produced in Crushing, Sq Cm per G	Surface Produced Per Unit of Work, Sq Cm per Kg Cm	Crushing Resistance, Kg Cm per Sq Cm
Quartz						
(Milky vein)						
F	12.8	127.8	202.3	74.5	5.81	0.172
G	11.54	127.8	203.0	75.2	6.51	0.154
H	29.1	43.1	210.6	167.5	5.76	0.173
I	26.5	43.1	198.5	155.4	5.87	0.170
J	62.1	14.1	335.9	294.2	5.18	0.193
K	51.6	43.1	337.3	321.8	5.71	0.175
L	61.8	14.1	348.0	333.9	5.40	0.185
M	24.4	43.1	179.5	136.4	5.60	0.179
N	32.5	74.1	249.4	175.3	5.40	0.185
					Avg 5.69	0.176
(Crystalline)						
Cr-1	7.97	1.45	63.75	62.3	7.81	0.128
Cr-2	37.14	40.7	266.0	225.3	6.06	0.165
					Avg 6.94	0.147
Calcite						
C-1	23.3	23.1	296.0	272.9	11.70	0.0855
C-2	17.59	23.1	200.7	177.6	10.10	0.0995
C-3	40.06	23.1	470.5	447.4	11.17	0.0900
C-4	33.5	58.3	383.5	325.2	9.72	0.103
					Avg 10.67	0.092
Fluorite						
F-1	41.92	14	697.0	683.0	16.3	0.0613
F-2	28.1	14	421.2	407.2	14.5	0.0690
F-3	17.37	14	496.2	272.2	15.67	0.0638
					Avg 15.49	0.0647
Labradorite						
L-1	23.63	43	192.0	149.0	6.31	0.159
L-2	38.55	43	274.3	231.3	6.00	0.167
L-3	51.81	43	351.5	308.5	5.95	0.168
L-4	58.78	43	376.0	333.0	5.67	0.177
					Avg 5.98	0.168
Sphalerite						
S-1	26.07	6.2	278.3	272.1	10.08	0.099
S-2	50.54	6.2	463.5	457.3	9.06	0.110
					Avg 9.57	0.105
Apatite						
A-1	41.18	70.7	389.3	318.6	7.74	0.129
A-2	55.6	70.7	492.2	421.5	7.58	0.132
					Avg 7.66	0.1307
Topaz						
T-1	53.3	1.1	268.2	267.1	5.00	0.200
Glass						
G-1	20.85	5.0	141.1	136.1	6.58	0.152
G-2	36.75	5.0	131.6	226.6	6.16	0.163
G-3	56.83	5.0	329.0	319.0	5.61	0.178
G-4	74.17	5.0	436.0	431.0	5.81	0.172
G-5	46.70	5.0	287.0	282.0	6.06	0.165
					Avg 6.05	0.166

Table II. Summary of Results Using Drop-Weight Crusher and Gas Adsorption

Material and Experiment No.	Net Work to Crushing, Kg Cm per G	Gas Adsorption Data			Air Permeability Data			Ratio of Final Surfaces, Adsorption to Permeability
		Initial Surface, Sq Cm	Final Surface, Sq Cm	New Surface, Sq Cm Per G	Initial Surface, Sq Cm	Final Surface, Sq Cm	New Surface, Sq Cm Per G	
Crystalline quartz								
Q-1	24.4	1350	8130	313	635	3960	153.5	2.05
Q-3	28.3	1080	6830	340	516	3070	155.0	2.22
Q-4	27.0	1130	6550	323	522	3450	174.2	1.90
Q-5	14.1	435	1230	158.8				
Q-8	134.1	385	6250	1170	155	3320	636	1.88
Q-9	191.4	404	8370	1590	155	4410	848	1.90
Q-10	37.1	364	2620	450				
Q-11	23.5	370	1940	315				
Q-12	231.0	370	11990	2320	155	6250	1230	1.92
Q-13	58.3	372	3800	683	155	2150	398	1.77
Q-14	22.0	387	1820	287				
Q-15	23.6	710	3750	305				
Q-16	21.4	3220	4770	309				
Q-17	21.1	1410	2750	268				
Q-18	98.2	3200	8140	985				
Massive quartz								
HQ-1	76.8	656	4630	794	220	2640	484	1.76
HQ-2	154.5	680	7930	1450	220	4230	800	1.87
Milky vein quartz								
MQ-1	95.5	1075	6300	1100	150	3200	610	1.97
MQ-2	53.2	1075	4330	660	150	2034	377	2.13

Experiment No.	Net Work to Crushing, Kg Cm per G	Initial Surface, Sq Cm	Final Surface, Sq Cm	New Surface, Sq Cm per G
CQ-12	177.0	370	12270	2350
CQ-13	158.0	340	9870	2100
CQ-14	107.8	370	8105	1550
CQ-15	65.9	370	5200	965
CQ-16	30.6	370	2215	370
CQ-17	8.9	330	1035	140
CQ-18	131.9	370	10000	1925
CQ-19	170.6	380	12940	2500

Table III. Summary Data Using Slow Compression Crusher and Gas Adsorption

of this energy is proportional to the net energy input, it appears logical to assume that plastic deformation caused the curvature in the relationship between new surface and energy input.

Adsorption Area Measurement: Since there was always some doubt as to what surface was being measured by the permeability method, it was decided to build a gas adsorption apparatus to use for measuring surface areas, see Fig. 8. *A* and *B* are the diffusion and the mechanical pumps, *C* and *D* are McLeod gages, *E* is the sample tube, *F* is a vapor pressure bulb, *G* is a gas reservoir, and *H* is a cold trap. The nos. 1 through 5 designate mercury cut-

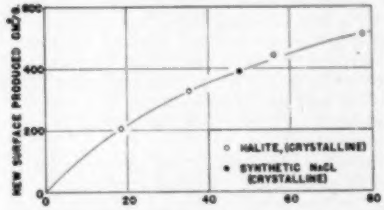

Fig. 7—Energy-new surface relation for impact crushing of salt.

offs used to isolate parts of the system. A complete description of this apparatus and its use is given in reference 33. Suffice it to say that ethane was the adsorbent at the temperature of liquid oxygen and that adsorption was carried out in the pressure range of 0.5 to 3.5 microns. The Brunauer, Emmett, Teller method was used to calculate surface areas.

A series of experiments was conducted with 5 to 20-g samples of quartz in which the crushing was accomplished with the drop-weight crusher and the areas were measured by the gas adsorption method and, in some cases, also by the air permeability method. The results of this work are tabulated in Table II and plotted in Fig. 9. Although Fig. 9 shows a curved relationship between the new surface and the net energy input instead of the straight line relationship previously obtained, there is no discrepancy since the previous work covered such a small range that the curvature was not evident. Fig. 10 is a plot of the areas of some samples as measured by gas adsorption vs. the areas of the same samples as determined by air permeability. It can be seen that the ratio of adsorption area to permeability area is fairly constant over the range studied and is equal to about 1.9. In some later work with single quartz particles with ground surfaces, the ratio of adsorption area to geometric area was as large as 17.

Slow Compression Crusher: In an attempt to obtain a more accurate measure of the actual energy input to the material, a series of experiments was

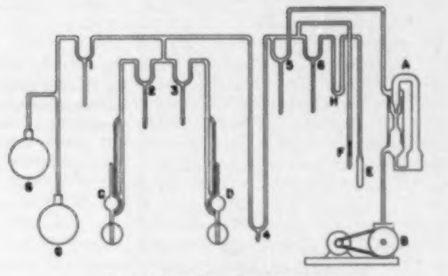

Fig. 8—Gas adsorption apparatus.

conducted[34] in which the crushing of quartz was accomplished by slow compression in a mortar between the platens of a small hydraulic press. Fig. 11 shows the crushing assembly used. The material is contained in a steel mortar *A* which is placed on the ram of the press as shown. Two dial gages *B* attached to a steel cylinder between the top of the mortar plunger and the top platen of the press serve to measure the displacement of the applied force. Since the energy imparted to an empty cylinder with the application of a load was found to be elastically returned with the release of the load, the net displacement noted during a crushing experiment represented energy imparted to the material being crushed. The actual energy imparted to the material was computed from the area under the curve obtained by plotting displacement vs. applied force. A typical plot is shown in Fig. 12.

In the first series of experiments using slow compression crushing, 5-g samples of 10/14 mesh crystalline quartz were used, and the areas were measured by the gas adsorption technique previously described. A summary of the data is given in Table III, and Fig. 13 is a plot of the new surface produced vs. the net energy input. A straight line relationship was again established. The discrepancy between this work and the previous work with the drop-weight crusher will be explained later.

Crushing of Single Particles of Quartz: All of the work with various minerals and the different methods of crushing, energy determination, and area measurement, accumulated valuable data but did not seem to give any explanation as to the basic mechanism of crushing. Therefore, a series of experiments was conducted in which single particles of crystalline quartz were crushed by the slow compression

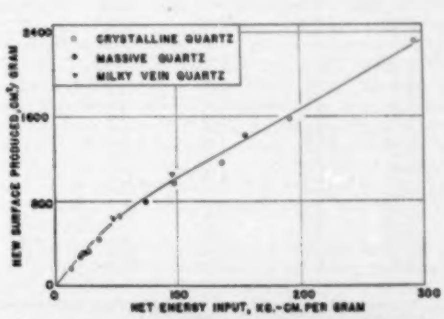

Fig. 9—Energy-new surface relationship for impact crushing of quartz. Surfaces measured by gas adsorption.

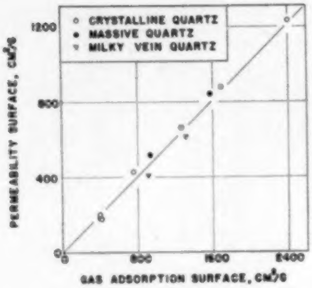

Fig. 10—Plot of surface measured by air permeability vs. surface measured by gas adsorption. Crushed quartz samples.

method, and the areas were measured by gas adsorption.[5] The quartz specimens usually weighed between 1 and 2 g. In some experiments a natural crystal was used while in others the crystal was ground to a cubical shape, but the crushing force was always applied to two 10$\bar{1}$0 faces of the crystal. When this material was subjected to the compressive force of the press, a definite displacement could be measured up to the point of fracture. Some cracking or spalling of the specimen was audible during the force application, and the final crushing usually occurred with explosive violence, including a sharp report and the shattering of the original particle into very small pieces as shown in Fig. 14.

Since the return of the elastic energy imparted to the crushing assembly during compression could not be measured after fracture occurred as with the multiple particles, a correction was made for this. Therefore, the net energy for crushing was taken as the area under the force-displacement curve where the displacement is the measured displacement minus the calibrated displacement of the crusher assembly. Fig. 15 shows the general nature of these curves.

Table IV gives a summary of the data for the single particle experiments and Figs. 16 and 17 are plots of these data as the new surface formed per unit of energy input vs. the average energy concentration. For comparison, the plots also show the previous results obtained with multiple particles. It has been pointed out to the authors that an equation for the data in Table IV and the curve in Fig. 16 can be written in the following form:

$$S_n = 189/E_c^{\text{[illegible]}}$$

where S_n is the new surface produced per unit of energy input in sq cm per kg-cm and E_c is the energy concentration in kg-cm per g.

If the theoretically calculated surface energy of a material based on interatomic forces is taken as the criterion of the energy necessary to produce new surface, the energy used in crushing multiple particles in the drop-weight apparatus or the slow-compression assembly is found to be about 50 to 100 times the theoretical amount. Therefore, if these surface energy figures are accepted, either present day methods of crushing are very inefficient in the utilization of the energy imparted to the material or the crushing process is inherently an inefficient process.

The results described for single particles of quartz indicate that the efficiency of the crushing process can, in some cases, be as much as 19, and perhaps even more times, greater than the usual value obtained for the crushing of multiple particles. However, even under these conditions of apparently high efficiencies, the actual efficiency is only about 25 pct when based on a value of 980 erg per sq cm for the surface energy of quartz. This value is an arbitrary one based on several theoretical values available.[5] It would appear that a large amount of the energy in any crushing operation is perhaps necessarily lost, largely as heat, without producing any new surface. Fahrenwald, et al[18] substantiated the earlier work of Cook[10] by demonstrating the actuality of a heat loss of 75 to 94 pct in a ball mill, but the necessity of such a loss has not yet been clearly established.

With only 1 to 2 pct of the energy input for the crushing of multiple particles appearing as surface energy in the crushed product, it seems quite improbable that a constant relationship could occur over any appreciable range between the energy input and the new surface produced. The straight line relationships which have been found experimentally are believed to be the result of a statistical average for the large number of particles crushed. The work with the single particles of quartz has shown that a variation of as much as 1500 pct can be found for this relationship between any two particles, but an average value would be expected, and apparently was obtained, when 500 or more

Table IV. Summary of Data for Crushing of Single Particles of Quartz by Slow Compression

	Particle Size								
Experiment No.	Cross-Section,[a] Sq Cm	Thickness,[b] Cm	Total Energy Input, Kg Cm	Average Energy Concentration, Kg Cm Per G	Initial Surface, Sq Cm	Final Surface, Sq Cm	New Surface, Sq Cm Per G	New Surface Per Unit of Energy Input, Sq Cm Per Kg Cm	Theoretical Efficiency, Pct[d]
SQ-10	0.60	0.60	1.2	0.6	124	430	159	265	26.5
SQ-6	0.47	0.55	2.7	4.2	43(c)	537	750	180	18.0
SQ-8	0.28	1.53	0.3	0.3	64(c)	118	50	166	16.6
SQ-15	0.66	0.87	2.4	1.6	78	415	218	139	13.9
SQ-17	0.49	1.10	2.0	1.3	69(c)	307	173	115	11.5
SQ-13	0.36	0.60	2.1	1.7	102	326	180	106	10.6
SQ-16	0.36	0.90	2.0	2.3	49	243	226	98	9.8
SQ-4	0.78	1.20	28.8	11.0	70(c)	2295	849	77	7.7
SQ-21	0.47	1.15	6.8	4.8	69(c)	538	330	69	6.9
SQ-9	0.83	0.55	15.4	13.6	70(c)	1140	940	69	6.9
SQ-14	2.12	1.50	35.9	2.5	344(c)	2470	150	60	6.0
SQ-20	0.43	1.04	12.3	10.4	61(c)	771	603	58	5.8
SQ-12	0.85	0.65	14.5	6.1	94	884	335	55	5.5
SQ-18	0.46	1.26	7.4	4.9	74(c)	466	261	53	5.3
SQ-11	0.46	0.50	9.8	6.1	101	496	245	40	4.0
SQ-19	0.39	1.10	20.0	17.2	60(c)	604	466	27	2.7
SQ-7	0.70	0.55	43.0	44.6	55	789	761	17	1.7

[a] Average area of two 10$\bar{1}$0 faces in contact with mortar bottom and plunger.
[b] Measured in direction of force application.
[c] Area calculated from geometric area and surface factor of 17.
[d] Based on value of 980 ergs per sq cm for the surface energy of quartz.

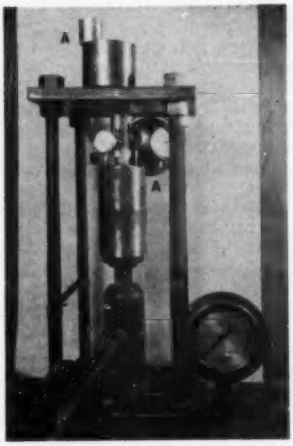

Fig. 11—Crushing assembly.
A—Mortar and plunger.
B—Dial gage assembly.

particles were used in the multiple particle experiments.

A direct comparison of the results for the drop-weight method of crushing and energy measurement, and the results obtained with the slow compression, force-displacement method can be made from the results shown in Figs. 9 and 13 and replotted together in Fig. 18. Although experimental evidence is lacking, the difference in results is attributed to a change in the usage of the energy imparted to the bed material as crushing proceeds in the drop-weight machine. In a proposed explanation it is assumed that the average stress concentration in the material with the drop-weight machine is a function of the energy input for each drop of the ball, and that a critical stress concentration is necessary before fracture will occur. With slow compression, the average stress concentration is independent of the energy input because the maximum force was kept constant. With the first drop of the ball in the drop-weight machine, the energy imparted to the bed of material is quite efficiently used to produce new surface because the average stress concentration in the bed of particles is relatively high and many particles reach the critical stress concentration necessary for fracture. With many successive drops of the ball, the energy input to the bed of particles decreases for a single drop of the ball by as much as 40 to 50 pct, and the average stress concentration is therefore less, if it is assumed that the stress concentration is a function of the energy input. As a result, fewer particles attain the critical stress concentration necessary for fracture, and more energy is lost without producing any new surface. This results in a successively lower value of new surface produced per unit of energy input as crushing proceeds in the drop-weight machine. The original work[illegible] using the drop-weight crusher showed that the energy input per drop of the ball had no effect on the final relationship between the new surface produced and the energy input but this conclusion was based on a relatively low energy input where the effect also is not very noticable in the present work.

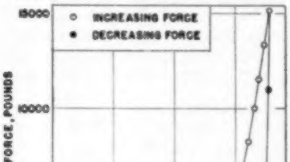

Fig. 12 — Typical plot of applied force vs. plunger movement for slow compression crushing.

Fig. 19 illustrates the differences in the areas of crushed quartz obtained by Gross and Zimmerley using the dissolution method and by the authors using the gas adsorption technique. As noted above, the curved relationship shown, curve *D*, is attributed to a change in the crushing process and not to the area measurement. Crystalline quartz was used in establishing this curved relationship except for two experiments, which were performed using a milky quartz believed to come from the same vein as that used by Gross and Zimmerley. It was found that the two types of quartz gave the same results. Since the same type of crusher and energy measurement was used in determining the relationships in curves *A*, *B*, and *D*, the results should be directly comparable. Gross and Zimmerley, curve *A*, obtained values of new surface by the dissolution method which were about 25 pct higher than those obtained by the authors in the same range using gas adsorption, curves C and *D*. This difference could be caused by a lack of consideration for the influence of the very fine material in the dissolution method as proposed by Hancock.[illegible] The latter has recalculated some of Gross and Zimmerley's data and obtained a value of new surface only 50 pct of that originally obtained as shown by curve *B*.

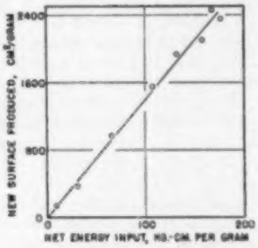

Fig. 13—Energy-new surface relationship for slow compression of crystalline quartz. Surfaces measured by gas adsorption.

It is significant that in the single crystal experiments the amount of new surface formed per unit of energy input was always greater than in the multiple particle experiments. At the lower average energy concentrations this ratio was as high as 19 to 1. Although the data for the single crystal experiments are quite scattered and group themselves most frequently around an average energy concentration near 5 kg-cm per g, a relationship is shown between the new surface formed per unit of energy input and the average energy concentration at fracture. One explanation of this relationship makes use of the concept of a critical stress value to produce fracture.

If the crushing takes place at a low average energy concentration, it is postulated that there must be a stress concentration at one or relatively few points. When fracture occurs, it originates at these few points of stress concentration, and each fracture can proceed to the edge of the crystal with a relatively

Fig. 14 — Single quartz crystals after crushing, X1.5. Top —experiment SQ-17. Bottom —experiment SQ-20.

large production of surface. Relatively few particles are formed, as shown in Fig. 14. The energy in the piece, other than at these few points of stress concentration, is lost as sound, vibration, etc. without any surface production.

When fracture occurs at a high average energy concentration, fractures are postulated to originate in rapid succession at large number of points, and a relatively large number of particles are formed, see Fig. 14. In this case, the critical stress concentration may be attained at a large number of points in rapid succession because of the high energy concentration and the irregularities or flaws on the immediately previous fracture surfaces. Now, instead of proceeding to the edge of the piece, each fracture proceeds only a short distance, with a corresponding small surface production, when it is stopped by the free surface of another fracture. Since the average energy concentration at fracture is higher than in the previous case, more energy is lost without surface production. The net result is a smaller production of new surface per unit of energy input as the average energy concentration increases because of a smaller surface production by each fracture and a greater loss of energy as a result of the higher energy concentration.

The theoretical efficiencies obtained with the experimental crushing machines described in the introduction are of the order of 1 to 2 pct. On the same basis (using a value of 980 erg per sq cm for the surface energy of quartz) the efficiencies obtained by the authors for the crushing of quartz

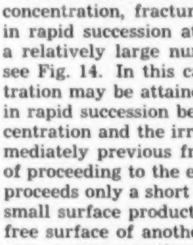

Fig. 15 — Force-displacement curves for crushing of three different quartz particles.

were 1.4 pct for multiple particles and from 1.7 pct to as high as 26.5 pct for single particles, with an average of 9.4 pct for the single particle experiments performed. Since the average value obtained for the single particle experiments is about seven times the value obtained in multiple particle crushing, a potential opportunity for considerable improvement in the design and operation of crushing machines is indicated.

As a consequence of the wide variation in the performance of individual particles, whether caused by variations in size or number of flaws, imperfections of contacts, some statistical or energy jump or other effects, and despite their demonstration for many cases of straight line relationships between energy input and new surface produced for the crushing of multiple particles in a bed, the authors feel that the law proposed by Rittinger is not a basic concept in crushing and should not be used as such. It is rather the result of a statistical average effect in crushing many particles and probably need be used with caution and with regard to the conditions of the experiment and the material. The curved relation for single particles shown in Fig. 16 demonstrates a much more fundamental phenomenon of crushing. Fig. 18, in turn, illustrates the importance of the conditions of the experiment and Fig. 7 shows the importance of plastic deformation in some materials. Rittinger's

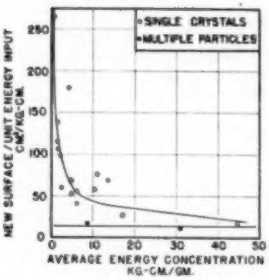

Fig. 16 — Relationship between new surface per unit of energy input and average energy concentration in crushing crystalline quartz.

law is probably too simple a concept to encompass a complicated phenomenon and may very well be a particularly desirable base or starting point for the development of a full understanding of crushing. Present studies in this laboratory are therefore pointed towards further experiments on single crystals and towards investigation of such intensive variables as temperature.

Acknowledgment

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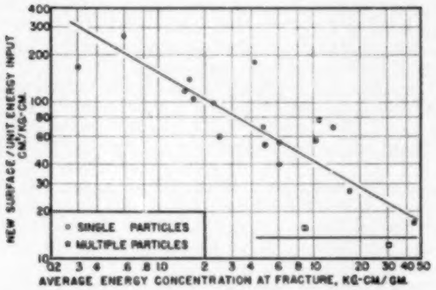

Fig. 17—Log plot of data in Fig. 16.

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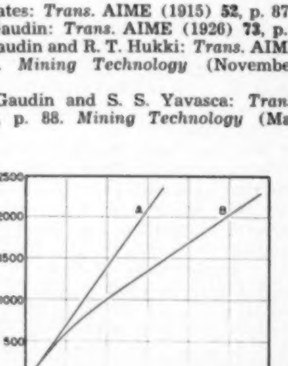

Fig. 18—Comparison of energy-new surface relationship for quartz with crushing by slow compression method and by drop-weight method.[24]
A—Slow compression, areas by gas adsorption.
B—Drop-weight, areas by gas adsorption.

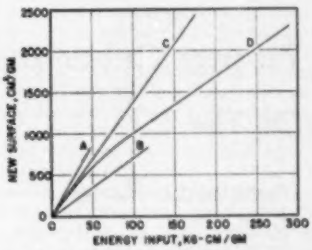

Fig. 19—Energy-surface relationship for the crushing of quartz by various investigators.
A—Drop-weight, dissolution.[23] B—Same as A, corrected by Hancock.[30]. C—Slow compression, gas adsorption.[24] D—Drop-weight, gas adsorption.[24]

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Tungsten Carbide Drilling On The Marquette Range

by A. Eugene Lillstrom

IN the development of iron mines and production of iron ore from the Marquette range, drilling blastholes is an important phase of the mining cycle. The ground drilled in ore production can be classified into two main categories, soft hematite and hard hematite or magnetite. Within these categories the material exhibits a wide range of penetrability by percussion drills.

Development work encounters various types of rock. Slate and altered basic intrusives constitute the softer types commonly encountered. Harder materials are represented mainly by greywacke, quartzite, iron formation, and diorite.

Prior to the first tungsten carbide trials in late 1947 and early 1948, hard-rock and ore drilling was done with steel jackbits starting at 2¼-in. diam. These were reconditioned by hot milling. Automatic or handcrank 3½-in. drifters were employed, mounted on Jumbos, posts and arms, or tripods, depending upon the working place. With the exception of shaft sinking jobs where 55-lb sinker machines were and still are used with 1-in. quarter octagon steel, the other production and development mining utilized 1¼-in. round and Leyner-lugged steel. The following properties have been selected as typical examples wherein carbide bit applications have proved economical. The Mather mine "A" and "B" shafts and Cleveland-Cliffs Iron Co. mines are soft ore mines where insert bits are used in rock development only. The Greenwood mine, Inland Steel Co., Champion mine, North Range Mining Co., and Cliffs shaft mine, Cleveland-Cliffs Iron Co., are hard ore mines where all drilling is done with tungsten carbide bits.

Mather Mine "A" Shaft

In the Mather mine "A" shaft and other soft ore properties where only rock development work is done with the tungsten carbide bits, several types and makes of bits have been tried since early 1948.

The greatest proportion of failures have been at the connection end, although the early trials with the 13 Series Carset 1½-in. bit used in conjunction with 3½-in. automatic-feed drifters, showed an equal amount of shattered inserts. To combat this shattering, the 3½-in. drifters were replaced by 3-in. drifters, thus eliminating, for the most part, insert failures. However, the attachment end of the rod continued to be the main source of trouble. The greatest amount of failure was in the stud or at the upset section approximately 2 in. behind the drive shoulder of the rod. Heat treatment was changed several times as well as the composition of the alloy studs. Since this failed to correct the trouble, a decision was made to change to a heavier attachment section. Timken 1½-in., type M, bits were then employed and showed an exceptional improvement. The rods are discarded when the thread contour shows sharpening or wear on the shoulder. It was also learned that the Timken insert did not show as rapid gage and cutting edge wear as did competitive makes, and footage per use increased by approximately 50 pct.

Prior to the Timken trials the average life per bit at the Mather mine "A" shaft on 6-ft change chain-feed drifters was 500 ft, and the rod life at the connection end was 50 ft. The Timken bit with chrome-plated thread averaged 1200 ft, and rod life increased to as much as 500 ft. However, the life of the connection end was much better on shorter length drill rods or in places where machines with 34-in. change were used.

The bit thread continued to be the point of ultimate failure with thread strippage, constituting the cause for discard of bits. In one of the new development headings, harder rock was encountered for approximately 800 ft, dropping the life per bit to a low of 90 ft with shank and thread life of rods dropping to approximately 125 ft average.

The stripped bits were then welded to the rods, increasing the life per bit by 75 to 100 pct. The rod transportation for main level development was not a problem so intraset rods were tried. Intraset rods have tungsten carbide inserts set into the rods proper by the manufacturer and can be obtained with chisel or four point bits. This type of rod eliminates the need for any connection and the steel being a special alloy will show more feet drilled per rod. The first trial was made with eight rods, and final results averaged 350 ft per rod, six of the rods worked the life of the bit end, and two broke shanks at less than 50 ft.

The preceding example showed a considerable improvement, so additional steel of the same type was purchased, but its use has been limited to main level drifting only, because of the handling problem involved in transportation of the complete rod to mine shops for resharpening.

Further trials are being made on improving the life per detachable bit by chrome plating. To date, the chrome plating shows an improvement of approximately 100 pct. However, final results will not be known until the present long term trials have been completed.

Mather Mine "B" Shaft

In November 1947, tungsten carbide bits were first tried at the Mather mine "B" shaft. The use of 1½-in. Carset 13 Series bits, for drilling the 72-hole, 7-ft shaft round, decreased the drilling time from an average of 4½ hr per round required with steel bits, to 2 hr with insert bits. The best drilling time for

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the tungsten carbide bits on an 8-ft round was 50 min, and it never exceeded 3½ hr. The maximum drilling time for steel bits was 11 hr, using 1700 jackbits for a 4-ft round. The steel bits averaged four uses per bit for a total of 6.2 ft per bit, whereas the tungsten-carbide bits averaged seven uses with total average of 135 ft drilled per bit.

The rods and bits were inspected at the shop before every drilling period.

The bits were gaged, and if found to have lost more than 1/16 in., were removed. The threads were also checked, and if sharpening of the thread contour was evident, the bits were discarded from shaft use. However, in development other than shaft sinking, these bits were used again, thus attaining an additional 50 pct life in feet per bit before final discard.

In rock development where Jumbo-mounted, 3-in. drifters were employed, the 1½-in. 13 Series bit did not prove satisfactory. The attachment end failures duplicated those explained in the example for the Mather mine "A" shaft. The threaded rod with 115 Series Carset special hard insert bit was tried and proved to be 100 pct better in feet per bit and approximately 500 pct better in feet per rod connection end. The entire mine was converted to 115 series with a result of 300 to 500 ft per bit, varying with the nature of the rock. Further experimentation with other types of bits or connections has been discontinued, at least temporarily, in favor of the standardization effected.

Greenwood Mine

Tungsten carbide trials at the Greenwood mine were started early in 1948 using various competitive types and makes of bits in conjunction with jackleg-mounted jackhammers. For drift work, 2⅝-in. jackhammers were used attached to pneumatic reverse feeds mounted on either Jumbos, or posts and arms. They also experienced similar connection end failures. By changing to a slower rotation on the up stroke, the life of the bit was improved by approximately 50 pct. Strippage continued to be the principal problem. Chrome plating of the bit threads was tried, with improvement, but later approximately the same results were obtained by turning the rod thread 0.002 in. oversize.

The results of the early trials are as follows: 1—Type A bit footage, drilled 1948, 55,490 ft; average per bit, 192 ft. 2—Type B bit footage, drilled 1948, 51,521 ft; average per bit, 163 ft. 3—Type A bit footage, drilled 1949, 127,000 ft; average per bit, 291 ft.

The total mine average for the year 1950 is 145 ft per bit with type A and C bits. The lower footage figure is attributed to the additional hard rock work encountered, which is charged for approximately 50 pct of the total bits for the year. The comparative figures are as follows: 1—Jumbo drifting in hard rock and jasper, 56 ft per bit; 2—Raise development, hard hematite, 259 ft per bit; 3—Stope development, hard hematite, 640 ft per bit; 4—Stope development, hard magnetite, 376 ft per bit.

The penetration speed averages 10 in. per min, varying from 6 in. to 15 in.

The increase in feet per bit drilled for the years 1949 and 1950 over that of 1948 is credited to miner training which included: 1—Insistence on the tightening of bits on rods with Stillson wrenches after each use. 2—Instruction that the machine should not be turned on until the bit is tight against bottom of hole when changing to longer lengths. 3—Orders that bits be returned for reconditioning when 3/32 in. wear shows on cutting edge. 4—The use of 0.005 in. undersize ring gages issued to the miner to control rod discard. 5—The use of ring gages by shift bosses to double check the miners' rods at least once a day.

The footage per bit in stopes and development drifts is quite satisfactory. The greatest amount of discard is caused by insert wear. Strippage accounts for approximately 10 pct of the discarded bits.

Raise development, however, in 1949, continued to keep the mine average per bit at a lower figure, because it is very difficult to keep a stoper machine tight to the end of the hole. A special stoper was made in the mine shops, with cylinder bore of the pneumatic leg increased in size from 2⅞ in. to 3⅛ in. The results to date show exceptional improvement, and the larger leg is now standard equipment for raise work in the Greenwood mine. However, with the added push on the leg it is necessary to use greater care in the construction of the drilling stages to compensate for the additional pressure.

Champion Mine

The Champion mine was reopened in April of 1948, and the first tungsten carbide bits tried were the 1¾-in., 15 Series Carset stud connection and 1½-in., 13 Series Carset stud connection. The average life per bit connection end was 150 ft using the miner training and precautions listed previously for the Greenwood mine. Next the 15 Series 1⅝-in., Carset bit was tried. The mine average was raised to 200 ft per bit, and an additional 50 ft was gained by welding the stripped bits to rods, thus increasing the final mine average to 250 ft per bit.

The Champion mine stopes are generally very near to the shaft and nipping is not a problem. Rods were brought to surface each day to be checked for wear and signs of fatigue. Because of this fact the Intraset rod was tried.

Using 4-ft rods with 1⅜-in. bit ends and 8-ft rods with 1 5/16-in. bit ends, results were as follows: Total drilled in year 1950, 202,985 ft; total average per machine shift, 120 ft; total per rod, 319 ft.; total cost per ft drilled exclusive of drillers labor, $0.0532.

The footage per man shift and inches per minute has increased approximately 10 pct through this use of Intraset rods, which is attributed to the smaller diameter holes and the elimination of the threaded bit end.

Down stroke rotation has been tried, but because of the nature of the ground, penetration speed dropped approximately 10 pct, thus making it impractical. One in 40 up stroke is now being used as standard rotation with penetration speed of 12 in. to 18 in. per min.

Cliffs Shaft Mine

Tungsten-carbide bit drilling was started at the Cliffs shaft mine in February 1948, used in conjunction with 3½-in. drifters and jackhammers with jackleg feeds. After a short trial period, it was learned that the available tungsten-carbide bits could not be used economically with the large 3½-in. handcrank, tripod-mounted drifter. Therefore, the experiments for the most part were limited to the use of the bits with 2½-in. and 2⅝-in. cylinder bore jackhammers mounted on a jackleg feed.

In the early tests, each bit was run to destruction without sharpening. Although the average total life during that experiment was 130 ft per bit, the practice was changed because the penetration rate which averages 8 in. per min dropped considerably after the first 60 to 75 ft of drilling, because of excessive

dulling of the bits, accompanied with complete insert failure. To remedy this condition, the bits were resharpened after a ⅛-in. flat appeared on the cutting edge. This was found to be the point to which the bit could be dulled before the insert would crack longitudinally, thereby causing destruction of the insert.

In spite of the care exercised in keeping bits sharp, a great deal of difficulty was experienced in the shattering of inserts during this early period. Finally, with the cooperation of the manufacturers and mine supervisors, this problem was nearly eliminated.

With the improvement of inserts to eliminate shattering, the total footage per bit increased considerably, bringing in a new problem, namely, connector thread strippage. This continued to be the main difficulty until early 1949.

The average insert bit cost per foot drilled for the year 1948 was $0.0898. Although slightly higher than the comparable steel bit costs, the obvious savings effected by small hole drilling made it advantageous to convert to tungsten carbide bits for general use and continue further experiments to improve the technique and bits.

At the beginning of 1949 the problems encountered with insert bit drilling were as follows: 1—Insert breakage approximately 25 pct caused by: (A) Longitudinal cracks apparently inherent in the available tungsten carbide inserts. (B) Complete breakage caused by use of too large machines. (C) Complete or partial destruction due to miners' own neglect. 2—Thread strippage 75 pct caused by: (A) Complete strippage where excessive tolerances were allowed by manufacturers on bit threads and rod stud threads. (B) Wear caused by tight and loose running of bit.

The efforts to eliminate thread strippage were as follows: 1—Thread contour gages were made to show the miner when he should discard his rod. This procedure resulted in culling 75 pct of the rods in use at that time. 2—Partly stripped bits were culled with hopes of preventing rod thread wear. The worn bits would, of course, wear a new rod and as a result, that worn rod helped wear or strip a new bit. 3—The shift bosses were given "no go" ring gages with which to check the rods at least once a day.

These three precautions resulted in improvement of bit appearance, footage drilled per man shift, and feet per bit; however, strippage continued to be the point of ultimate failure.

By pure circumstance, a down stroke rotation machine was placed in one of the contract headings. Exceptional bit life and no thread failure resulted. Because of this discovery, more machines were changed to down stroke rotation in those contract faces where excessive stripping of bits and rods was encountered. Bit life increased 50 to 100 pct with 100 pct elimination of thread strippage. Down stroke rotation is being used exclusively at the Cliffs shaft mine at the present time. However, a new problem was introduced, that of thread stud breakage. Often a bit must be discarded prematurely because the broken stud cannot be removed.

The change to minimize this difficulty was 1—slow down the down stroke rotation to a ratio of 1 in 40 which caused the point of ultimate failure to occur in the rod section, rather than the stud thread, and 2—discontinue the use of the stud 13 and 15 Series connection and convert to use of field-threaded rod for the 4-point, 115 Series 1⅝-in. Carset bit or the 113 Series 1⅜-in. Carset bit, which have proved most satisfactory for Cliffs shaft mine use.

Table I. Drilling Results

Year	Total Ft Drilled	Avg Ft Per Bit	Cost Per Ft Drilled
1948	125,218	125	.092
1949	281,148	170	.0713
1950	627,276	175	.077
January through March	149,748	188	0.0731
April through June	163,494	160	0.0841
July through September	147,250	112	0.1127
October through December	166,784	385	0.0405

During the first 9 months of 1950, drilling results steadily dropped in feet per bit, thus raising the cost per foot drilled. Table I gives the drilling results for 1948 through 1950. The 3-month periods for 1950 show an actual drop in footage per bit until the end of August, which is attributed to stud breakage in skirt of bit causing premature discard of the bits. The rise in feet per bit for the last 3 months is attributed to the discontinuance of the 13 Series stud and change over to the heretofore mentioned 115 Series threaded rods with 1⅝-in., 115 Series Carset bit.

In addition, the sharpening technique was changed. In place of the bench grinder, a J-3 Ingersoll-Rand Jackbit grinder was converted to insert bit sharpening. The use of the grinder provided more accurate control of gage and cutting angle. The wheel used is a Simons Worden White Co. 12x1x1¼-in. type, IK Grain, GC60-J14V14, or the Carborundum Co.'s silicon carbide wheel GC60-K11-VR and GC60-J8-VW. Other types of wheels have been tried, but the listed ones have proved most practical and satisfactory.

Summary

It is agreed by mine operators that future tungsten carbide drilling will of necessity be classed in two separate groups; the first being hard rock and hard ore drilling with speed of penetration from 3 in. to 15 in. per min and the second group, although limited at present on the Marquette range to 3-in. drifters, will use 3½-in. cylinder bore drifters for development in the future where speed is most important.

To gain greater penetration speed in hard materials of the first group, experiments with smaller diameter holes are now in progress using both detachable bits and rods with tungsten carbide inserts set into the rod proper. The indicated results are very favorable although final results will not be known for some time.

Experiments are also being made on the Marquette range with rotary drilling using tungsten carbide tipped auger steel. The results of the tests prove more satisfactory than the percussion type auger drilling now being used, and it is possible that the rotary drilling of soft hematite will be used much more extensively within the next year.

The various machine changes and technique precautions herein mentioned have improved, to a great extent, the footage per bit. However, it is the combined opinion of all users that better results will be realized as experiments are continued.

Acknowledgment

Gratitude is expressed to the Management and Personnel for their cooperation in making this paper possible.

Five Variable Flotation Tests Using Factorial Design

by Adrian C. Dorenfeld

Factorial design is a mathematical method of drawing valid conclusions from a series of tests made in a predetermined pattern. It is applied to flotation ore testing using, in this case, five variables, each varied twice, requiring a total of 32 tests. Representative conclusions are checked by repeat testing.

THE factorial design method of testing is particularly applicable to mill-scale plant tests because 1—the conclusions take into consideration all variabilities of the plant, 2—there are no subjective guesses as to the conclusions, 3—the number of tests necessary to check large numbers of variables are at a minimum, and 4—interactions between variables can be discerned. It is equally applicable to laboratory testing because once the variables to be investigated have been decided upon, as well as the number of levels for each variable, a well-trained assistant can run the tests, since the test work follows a predetermined mathematical pattern, leaving the conclusions to be found by the person in charge. The conclusion takes into consideration all errors of the tests. To illustrate the use of factorial design, a sample of copper porphyry ore was tested, the copper content of the batches not being uniform. Five variables were investigated, each at two levels, requiring only 32 tests. After suitable arithmetical analysis, conclusions for copper recovery and concentrate grade are arrived at. Four cases of these conclusions are checked by repeat testing.

Testing Methods

Reproducibility of flotation tests varies widely, depending on the experimenter, the process, the equipment, and probably a host of unknown and perhaps interacting variables. If the reproducibility of tests is good, let us say to 0.2 pct recovery of a certain metal, and the minimum difference between tests upon the variation of the interested variables is, say 1 pct, then simple averaging and inspection is ample in interpreting results.

However, often such a happy situation is not the case. This is particularly true in mill-scale operations where the ores are variable, and a host of minor operating fluctuations common to most milling, and familiar to all mill operators, is the normal state of affairs. Yet, mill scale tests are run, the results averaged, and conclusions drawn therefrom. The trustworthiness of such results should be measured by comparison to the experimental error—the measure of the degree of reproducibility of the results. This comparison is a test of significance. Thus, if daily results of recoveries average 80.0 pct, and a spread of 78.0 pct to 82.0 pct, is recorded, then, clearly, on changing mill operations, an 81 pct recovery cannot be construed as significant by using averages, since it is well within the previous test limits. By using rigorous statistical tests of significance, conclusions arrived at by inspection or simple averaging may be shown to be based on inadequate evidence or even wrong. In any event, a plant super-

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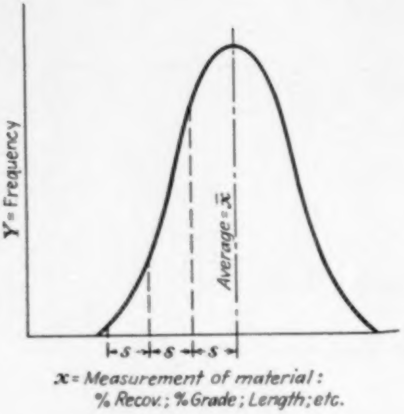

Fig. 1—Normal distribution curve.

intendent would like to know whether the recommendations, based on tests, can be shown to be scientifically sound, or are merely subjective guesses.

There are methods of testing statistically whether one method of operation is different from another. They are based on the use of relatively large numbers of tests and the variation of only one variable at a time. Therefore, to test as many as five variables a large number of tests would be required. The number of tests would be dependent on the reproducibility of results. However, there is a method, long used in agricultural experimentation and adopted for some years in the chemical industries, that uses a limited number of tests, executed in a mathematical pattern, and yields the following information: the significance of changes in relation to experimental error; the probable average and range of results; and whether there are interactions between variables or not. This method, called factorial design, makes use of the results of one test several times.

Definition of Terms

1—Normal distribution is a mathematical distribution made up of individuals perfectly described by knowing the average and standard deviation of these individuals, see Fig. 1.

2—Mean or average is the line which divides a distribution area in exactly equal parts, see Fig. 1. It is the moment of the data about the origin.[1]

3—Variance is a measure of variation of the data about the mean. It is the second moment of the data about the mean.[2]

4—Standard deviation is the square root of the variance.[4] It is given by the formula:

Table I. Tests on Copper Porphyry Ore

Factors	Symbol	Level 1	Level 2
Grind, mesh	G	−65	−200
pH	P	8.5	10.5
Xanthate, lb per ton	X	0.20	0.30
Sodium cyanide, lb per ton	C	0.0	0.10
Frother (cresylic acid), lb per ton	F	0.05	0.10

$$\text{standard deviation} = S = \sqrt{\frac{\Sigma X^2}{n} - \bar{X}^2} \qquad [1]$$

where: X^2 = the sum of the square of each individual term of the data

n = number of individuals in the data

$\bar{X}$ = the average of the data

5—Population or universe is the infinite number of tests that could be made using one particular physical condition. The greater the number of tests the closer the approach is to the population. The mean of the population is denoted by m, and the standard deviation by σ.

6—Sample is a finite number of tests, N, drawn at random from a population. Each sample will have its standard deviation, S, and average $\bar{X}$.

7—Significance level is the risk taken in being wrong. Thus a significance level of 5 pct means that in the long run the conclusion may be wrong five times out of 100 times. It is usually fixed by the economics of the situation. If large-scale costly additions would be necessary to install a new process, then a significance level of 1 pct or better would be justified; if the new process is merely a change in reagents, of substantially equal costs, then a significance level of perhaps 20 pct might be justified.

8—F distribution is the distribution of variances. Tables for this distribution exist.[4] If the following ratio:

$$F = \frac{NxSx^2/Nx\text{-}1}{NySy^2/Ny\text{-}1}$$

$$\nu 1 = Nx\text{-}1$$

$$\nu 2 = Ny\text{-}1$$

where

Nx = number of individuals in sample X

Table II. Tests on Copper Porphyry Ore

Test No.	G Mesh	P	X, Lb per Ton	C, Lb per Ton	F, Lb per Ton	Cu Recovery, Pct	Cu Concentrate Grade, Pct	Cu in Heads, Pct
1	65	8.5	0.2	0.0	0.05	21.21	5.65	0.92
2	65	10.5	0.2	0.0	0.05	70.58	17.40	0.95
3	65	8.5	0.3	0.0	0.05	49.33	6.19	0.88
4	65	10.5	0.3	0.0	0.05	60.65	12.47	0.83
5	200	8.5	0.2	0.0	0.05	17.28	1.95	0.86
6	200	10.5	0.2	0.0	0.05	85.43	14.75	1.03
7	200	8.5	0.3	0.0	0.05	21.82	3.67	0.82
8	200	10.5	0.3	0.0	0.05	75.30	8.73	0.89
9	65	8.5	0.2	0.10	0.05	27.26	6.90	0.93
10	65	10.5	0.2	0.10	0.05	64.78	19.62	0.99
11	65	8.5	0.3	0.10	0.05	60.87	11.00	0.88
12	65	10.5	0.3	0.10	0.05	66.67	13.58	0.78
13	200	8.5	0.2	0.10	0.05	25.14	5.82	0.84
14	200	10.5	0.2	0.10	0.05	75.69	10.13	0.87
15	200	8.5	0.3	0.10	0.05	39.83	8.52	0.92
16	200	10.5	0.3	0.10	0.05	69.89	10.67	0.91
17	65	8.5	0.2	0.0	0.10	57.65	6.58	0.95
18	65	10.5	0.2	0.0	0.10	76.35	11.86	0.92
19	65	8.5	0.3	0.0	0.10	47.64	4.53	0.85
20	65	10.5	0.3	0.0	0.10	74.40	8.14	0.70
21	200	8.5	0.2	0.0	0.10	39.99	2.35	0.70
22	200	10.5	0.2	0.0	0.10	77.97	4.72	0.58
23	200	8.5	0.3	0.0	0.10	49.64	3.51	0.75
24	200	10.5	0.3	0.0	0.10	75.93	6.71	0.73
25	65	8.5	0.2	0.10	0.10	62.87	18.91	0.99
26	65	10.5	0.2	0.10	0.10	74.84	17.40	0.92
27	65	8.5	0.3	0.10	0.10	60.61	17.32	0.91
28	65	10.5	0.3	0.10	0.10	72.07	13.90	0.79
29	200	8.5	0.2	0.10	0.10	70.66	8.14	0.76
30	200	10.5	0.2	0.10	0.10	82.61	8.81	0.80
31	200	8.5	0.3	0.10	0.10	62.29	10.40	0.73
32	200	10.5	0.3	0.10	0.10	86.12	12.66	0.98

Avg = 0.855
Standard Deviation, pct = 0.10

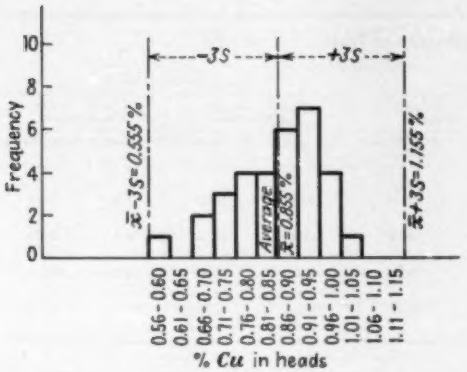

Fig. 2—Grade of heads, pct Cu, 32 tests.

Ny = number of individuals in sample Y

S^2x = variance of sample X

S^2y = variance of sample Y

$\nu 1$ = degree of freedom of X

$\nu 2$ = degree of freedom of Y

is smaller than the ratio in the F distribution table for the particular level of significance, then X and Y are samples drawn from the same population, and in the long run will be equal a greater percentage of the time than the level or significance used.[5]

Experimental

A sample of a copper porphyry ore was obtained and crushed to −10 mesh. The ore was not thoroughly mixed because a variable ore was wanted to simulate conditions in practical milling. One thousand gram charges of the ore were made up. The charges for testing were selected at random. Five variables were investigated at two levels. The results are given in Table I.

The mathematical pattern calls for 32 tests being run. The results of these tests are given in Table II.

Factorial design of an experiment is the planning of tests in such a way that the result of each test can be used several times. It is based on the premise that the results of the experiment are normally distributed, and therefore the equations for this distribution apply. From the Tchebycheff inequality, it has been shown that for total nonnormality, the greatest error possible at 2σ limits is 25 pct, if the normal distribution equations are applied to nonnormal distributions.[6] Further, Camp[7] and Meidell[8] have shown that if a distribution is such that it falls off rapidly from the mean, the greatest error possible is 11.1 pct, at 2σ limits, in applying the normal distribution equations. For normal distributions, the greatest error possible is 4.5 pct.[8]

Fig. 2 shows a graph of the distribution of copper content of the heads and of one series of concentrates. They fulfill the conditions of Camp-Meidell, and therefore the normal distribution equations can be used with confidence. Since the balance of the tests are similar in nature, then by analogy, their distributions will be similar. Hence, the method of factorial design can be safely applied.

For normal distributions, variances are addative.[9] Thus, if a series of tests is run varying only one variable, then the variance is S_1^2. Similarly the variance of variable S_2 is found, etc. The variances of all the tests is the sum of all the single variable variances S_1^2, S_2^2--S_n^2 or:

$$S^2 \text{ Total} = S_1^2 + S_2^2 + S_3^2 + \text{--}S_n^2 \qquad [2]$$

Another method for obtaining the total variance is to consider all the tests as one series and obtain it in the usual way as described in the definition of terms. If the tests are perfect, the variances will be equal. This is seldom the case. The difference between the two total variances is the variance of all unknown factors—it is the variance of the experimental error.[10, 11] From this fundamental concept, the method for testing component variances is built up. Further mathematical proofs are in the texts listed in the references.[1, 9] The mechanics of the analysis for the recovery of copper follows.

The mathematical pattern for testing is merely that all possible combinations of tests are run. In this case, for a five variable test at two levels each, it is 32 tests, see Table III.

Eq 1 showed that the standard deviation is:

$$S = \sqrt{\frac{\Sigma X^2}{n} - \overline{X}^2}$$

$$\text{or } S^2 = \frac{\Sigma X^2}{n} - \overline{X}^2$$

since the average $\overline{X}$, is the sum of all the individuals, divided by the number of such individuals, or $\overline{X} = \dfrac{\Sigma X}{n}$, we can write:

$$S^2 = \frac{X^2}{n} - \frac{(\Sigma X)^2}{(n)^2} \qquad [3]$$

multiplying by n, we get:

$$nS^2 = \Sigma X^2 - \frac{(\Sigma X)^2}{n} \qquad [4]$$

From the definition of terms, the F ratio $(NxSx^2/n\text{-}1) \div (NySy^2/n\text{-}1)$ is a test of significance whether sample x and y are randomly related or not, it is clear interest should be centered on the nS^2 terms. This is called the sum of squares. The $nS^2/n\text{-}1$ terms are the unbiased variances.[12] To obtain the sum of squares, eq 4 shows that it is necessary to square

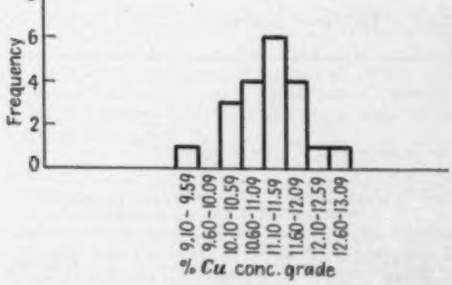

Fig. 3—Grade of concentrate, pct Cu, 20 repeat tests of test 32.

Table III. Copper Recoveries of Tests

		G_1 = 65 Mesh Grind				G_2 = 200 Mesh Grind			
		X_1 = 0.2 Lb per Ton Xanthate		X_2 = 0.3 Lb per Ton Xanthate		X_1 = 0.2 Lb per Ton Xanthate		X_2 = 0.3 Lb per Ton Xanthate	
		P_1 = pH8.5	P_2 = pH10.5	P_1 = pH8.5	P_2 = pH10.5	P_1 = pH8.5	P_2 = pH10.5	P_1 = pH8.5	P_2 = pH10.5
F_1 = 0.05 lb per ton	C_1 = 0.0 lb per ton NaCN	21.21	70.58	49.33	60.65	17.28	85.43	21.82	75.30
Frother	C_2 = 0.1 lb per ton NaCN	27.26	64.78	60.87	66.67	25.14	75.69	39.83	69.89
F_2 = 0.10 lb per ton	C_1 = 0.0 lb per ton NaCN	57.65	76.35	47.64	74.40	39.99	77.97	49.64	75.93
Frother	C_2 = 0.1 lb per ton NaCN	62.87	74.84	60.61	72.07	70.66	82.61	63.29	86.12

Table IV

$G_1 = G_2$

		X_1 = 0.2 Lb per Ton Xanthate		X_2 = 0.3 Lb per Ton Xanthate	
		P_1 = pH8.5	P_2 = pH10.5	P_1 = pH8.5	P_2 = pH10.5
F_1 = 0.05 lb per ton	C_1 = 0.0 lb per ton NaCN	38.49	156.01	71.15	135.95
Frother	C_2 = 0.1 lb per ton NaCN	52.40	140.47	100.70	136.56
F_2 = 0.10 lb per ton	C_1 = 0.0 lb per ton NaCN	97.64	154.32	97.28	150.33
Frother	C_2 = 0.1 lb per ton NaCN	133.53	157.45	123.90	158.19

each individual and add; subtract from this the square of the sum divided by total number of tests. This last, $\frac{(\Sigma X)^2}{n}$, is called the correction factor. It is actually part of the arithmetic in arriving at the variance. Similar calculations are carried out in arriving at the component variances. The mathematics used to separate the component variances from the total number of tests are described by P. G. Hoell.[13, 14]

The mechanics of using the test results of Table III follows: The sum of the tests are squared and divided by the number of tests:

$$\frac{(\Sigma X)^2}{n} = \frac{1{,}904.37^2}{32} = 113{,}332.03$$

This is the correction factor, (C.F.) already discussed. Each individual test result is squared and summed: $\Sigma X^2 = 125{,}734.63$. The total sum of squares is $\Sigma X^2 - \frac{(\Sigma X)^2}{n}$ or 12,402.60. Each variable is summed over in turn. This means that the assumption is made that in the case of grinding, the difference in results between 200 mesh grinding and 65 mesh grinding is no more than would be made if duplicate tests at each grind were made. Should this not be the case, then in the final analysis of variance, the variances of each component will be significantly greater than the experimental error variance. Yet, this preliminary assumption does not invalidate the final results.[15] Evidence obtained by repeat testing will be presented that this is physically the case. As an example, assume the results of 65 mesh grinding, G_1, are but an experimental variant of 200 mesh grinding, G_2, or $G_1 = G_2$. Then test $F_1\,C_1\,P_1\,Q_1\,G_1$ is an experimental variant of test $F_1\,C_1\,P_1\,Q_1\,G_2$ and can be added together. Thus Table IV is built up.

Similarly four other tables are built up, letting $X_1 = X_2$; $P_1 = P_2$; $C_1 = C_2$; $F_1 = F_2$.

The next step is to sum over the variables two at a time. This is done by taking each table as built up in Table IV, and assuming each variable to be equal, or as in Table IV, $X_1 = X_2$. Such a table is illustrated in Table V.

Similarly nine other such tables are computed. The next step is summing over the variables three at a time, or in this illustration, allowing $G_1 = G_2$; $X_1 = X_2$; $P_1 = P_2$. Table VI is thus computed:

Nine such tables can then be computed. The sum of squares of each component can now be obtained. The correction factor is known. The mechanics follow:

Table V

$G_1 = G_2$; $X_1 = X_2$

P_1 = pH8.5				P_2 = pH10.5			
C_1 = 0.0 Lb per Ton NaCN		C_2 = 0.1 Lb per Ton NaCN		C_1 = 0.0 Lb per Ton NaCN		C_2 = 0.1 Lb per Ton NaCN	
F_1 = 0.05 lb per ton Frother	F_2 = 0.10 lb per ton	F_1 = 0.05 lb per ton	F_2 = 0.1 lb per ton	F_1	F_2	F_1	F_2
109.64	194.92	153.10	257.43	291.96	304.65	277.03	315.64

Table VI

$$G_1 = G_2;\ X_1 = X_2;\ P_1 = P_2$$

	C_1 = 0.0 Lb per Ton NaCN	C_2 = 0.10 Lb per Ton NaCN	Totals
F_1 = 0.05 lb per ton Frother	401.60	430.13	831.73
F_2 = 0.10 lb per ton Frother	499.57	573.07	1072.64
Totals	901.17	1003.20	1904.37 Grand Total

Table VII. Variances for Cu Recoveries

Type Re-action	Source of Variance	Degrees of Freedom (n-1)	Sum of Squares (nS^2)	Variance $\frac{(nS^2)}{(n-1)}$
Main Effect	Grind = G	2−1 = 1	2.43	2.43
	Xanthate = X	2−1 = 1	59.82	59.82
	pH = P	2−1 = 1	7,026.76	7,026.76
	NaCN = C	2−1 = 1	325.33	325.33
	Frother = F	2−1 = 1	1,813.68	1,813.68
First Order Reaction	G x X	(2−1) (2−1) = 1	27.47	27.47
	G x P	(2−1) (2−1) = 1	515.12	515.12
	G x C	(2−1) (2−1) = 1	44.43	44.43
	G x F	(2−1) (2−1) = 1	29.54	29.54
	X x P	(2−1) (2−1) = 1	301.28	301.28
	X x C	(2−1) (2−1) = 1	23.20	23.20
	X x F	(2−1) (2−1) = 1	154.13	154.13
	P x C	(2−1) (2−1) = 1	377.50	377.50
	P x F	(2−1) (2−1) = 1	597.80	597.80
	C x F	(2−1) (2−1) = 1	63.19	63.19
All other Residual Interactions 10 third order 5 fourth order 1 residual		(10+5) + (2−1) (2−1) (2−1) (2−1)) (2−1) = 10+5+1 = 16	1,040.93	65.06
Total			12,402.60	

The sum of squares due to cyanide is:

$$\frac{\overline{901.17^2} + \overline{1{,}003.20^2}}{16} = 113{,}657.35$$

$$\begin{aligned} \text{Correction Factor} &= \underline{113{,}332.03} \\ \text{Sum of squares due to cyanide} &= 325.32 \end{aligned}$$

Similarly the sum of squares due to each factor is obtained.

There is always a possibility that the reaction between two variables is the factor responsible for the results. Physically, this is the case for cyanide and hydroxyl ion concentrations. It has been amply illustrated that the cyanide ion concentration is the governing factor in copper depression. This ion concentration can be varied by varying the pH. Yet the pH alone, within limits, has no bearing on copper depression. To obtain such interactions, each term of Table VI is squared and divided by the number of terms composing it. As an illustration, the sum of squares for the interaction between cyanide and frother would be:

$$\frac{\overline{401.60^2} + \overline{430.13^2} + \overline{499.57^2} + \overline{573.07^2}}{8} \quad \text{or} \quad 115{,}534.22.$$

Sum of squares due to cyanide—	325.32
Sum of squares due to frother—	1,813.68
Correction Factor	113,332.03
Sum of squares due to interaction of cyanide and frother	63.19

Similarly the other interactions are obtained. Second order interactions are also possible. These would be reactions among any three of the variables. There might also be third order interactions as well. However, in flotation testing, the possibility of interactions greater than first order is probably small, from a consideration of the chemistry involved. However, in poorly known processes, all interactions should be investigated. A complete set of variances of all interactions will be presented in an analysis of concentrate grade.

The variances shown in Table VII then were obtained.

Residual sum of squares = total sum of squares − main effect squares − first order squares.

All the first order interaction variances numerically less than the residual — *G x X, G x C, G x F, X x C, C x F*, are not significant. The experimental error is greater than these variations. Therefore, the variations of these interactions are a part of the experimental error.[18] The results of using the *F* dis-

Table VIII. F Test for First Order Interactions

Interaction	Variance	Interaction ÷ Residual	5 Pct Level of Significance from Tables
X x F	154.13	2.37	4.49
X x P	301.28	4.44	4.49
P x C	377.50	5.80	4.49
G x P	515.12	7.92	4.49
P x F	597.80	9.19	4.49

Table IX. Variances at P_1 and P_2 Levels
Two 4 Factor Experiments

Type Reaction	Source of Variance	Degrees of Freedom (n-1)	Sum of Squares (nS^2)	Variance $\frac{(nS^2)}{(n-1)}$
At P_1 Level (pH = 8.5).				
Main Effect	G = Grind	(2−1) = 1	223.42	223.42
	X = Xanthate	(2−1) = 1	314.79	314.79
	C = NaCN	(2−1) = 1	701.85	701.85
	F = Frother	(2−1) = 1	2246.99	2246.99
First Order Reactions	G x X	(2−1) (2−1) = 1	48.84	48.84
	G x C	(2−1) (2−1) = 1	74.01	74.01
	C x F	(2−1) (2−1) = 1	152.00	152.00
	X x C	(2−1) (2−1) = 1	2.54	2.54
	X x F	(2−1) (2−1) = 1	517.00	517.00
	C x F	(2−1) (2−1) = 1	23.60	22.60
Four Second Order Interactions + Residual	Residual	5	341.49	68.30
At P_2 Level (pH = 10.5)				
Main Effect	G	(2−1) = 1	294.12	294.12
	X	(2−1) = 1	46.31	46.31
	C	(2−1) = 1	0.07	0.07
	F	(2−1) = 1	164.48	164.48
First Order Interactions	G x X	(2−1) (2−1) = 1	50.18	50.18
	G x C	(2−1) (2−1) = 1	0.66	0.66
	G x F	(2−1) (2−1) = 1	21.77	21.77
	X x C	(2−1) (2−1) = 1	27.25	27.25
	X x F	(2−1) (2−1) = 1	29.83	29.83
	C x F	(2−1) (2−1) = 1	41.99	41.99
Four Second Order Interactions + Residual	Residual	5	52.06	10.41

Table X. F Test for First Order Interactions, P_1 Level

Interactions	Variance	Interaction Variance Residual ÷ Variances	5 Pct Level Value from Tables
$G \times C$	74.01	1.1	6.61
$C \times F$	152.60	2.2	6.61
$X \times F$	517.00	7.0	6.61

Table XI. First Order Reactions Grouped with Residual

Source of Variance	Degree of Freedom	Variance
G	1	223.42
X	1	314.79
C	1	701.85
F	1	2246.99
$X \times F$	1	517.00
Residual	10	64.22

tribution to test whether the variances greater than the residual are significant, are given in Table VIII.

From Table VIII, the only interaction not significant is that of $X \times F$—the other four reactions are significant. These four reactions have one common factor, pH, or P. This may mean that the interactions are significant merely because the main effect, P, is very large. To test this, P can be eliminated by breaking down the five factor experiment to two four factor experiments; one series of tests at P_1 level, (pH8.5) and the other series at P_2 level, (pH10.5). Table IX is then obtained.

Consider the results at the P_1 level. Only the following interactions are numerically greater than the residual, see Table X.

Only the $X \times F$ interaction is significant. The results of grouping the nonsignificant first order interactions with the residual are shown in Table XI.

The G and C main effects, to exist, must be significantly greater than the residual; the X and F effects must each be significantly greater than the $X \times F$ interaction, because the X and F main effects are each made up of the $X \times F$ variance plus the main effect variances.[17] Applying the F test for the above table, Table XII is obtained.

Therefore, the conclusion is that the X, F, and G effects do not exist. Variation of grind, xanthate or frother do not, by themselves, affect the results. Differences caused by changes in these physical variables are merely experimental errors. However, xanthate and frother changes affect each other; using 0.2 lb per ton xanthate and 0.05 lb per ton frother cannot be compared to 0.2 lb per ton xanthate and 0.10 lb per ton frother on the basis that

Table XII. F Test Applied for Table XI

Source of Variance	Source of Variance ÷ Residual Variance or the Interaction Variance	5 Pct Level from Tables
G	3.5	4.96
C	10.9	4.96
X	Numerically $< X \times F$	161.0
F	4.4	161.0

superior results of the latter are caused by the increase of frother—that frother acting only as a frother has increased the recovery of the metal. The tests show that at a pH of 8.5, the frother reacts with the xanthate, affecting the collection of copper. It is the reaction between the two that causes the results. Physically, this is explainable by previously published results and will be discussed in detail after the conclusions are given.

The only significant main effect is cyanide. The other effect is the xanthate-frother interaction. All tests that have the three variables in common: cyanide, xanthate, and frother, at any one level, can be pooled, because the other variable, grinding, has been shown to be nonsignificant. It does not affect the results more than by chance causes—the experimental error. It is immaterial whether grinding to 65 mesh or to 200 mesh is practiced. Any difference due to this change is within the error of results caused by ore variability, process variability, sampling variability, assay variability, etc. The conclusions are summarized in Table XIII.

Table XIII. Conclusions at pH8.5 Variability of Grinding Nonsignificant

Conclusion	Significant Variables, Lb per Ton: Xanthate	Frother	Cyanide	Test No. in Table II	Recovery Average Cu, Pct	Standard Deviation Cu, Pct
1	0.2	0.05	0.00	1,5	19.25	2.04
2	0.2	0.05	0.10	9,13	26.45	1.31
3	0.2	0.10	0.00	17,21	48.82	8.83
4	0.2	0.10	0.10	25,29	66.77	3.81
5	0.3	0.05	0.00	3,7	35.58	13.75
6	0.3	0.05	0.10	11,15	50.35	10.54
7	0.3	0.10	0.00	19,23	48.64	1.00
8	0.3	0.10	0.10	27,31	61.95	1.34

It will be noted that some of the conclusions yielded results with large standard deviations. These results did not have good reproducibility, particularly conclusions 5 and 6. These tests used 0.3 lb per ton xanthate and 0.05 lb per ton frother. The mathematics show an interaction occurred. Physically, these froths were scummy and exceptionally thin. Pulp showed throughout these froths. The reaction can be explained by the fact that a substantial part of the frother, cresylic acid, could have been attached to the sulphide surfaces, the xanthate serving as anchors, and therefore not leaving enough frother for frothing.[18] In conclusions 1 to 4, a smaller amount of xanthate was used, and therefore not as much cresylic acid could have been attached to the sulphide particles to leave enough cresylic acid for frothing purposes. Similarly, in conclusions 7 and 8, 0.1 lb per ton cresylic acid was used, or twice as much as in tests making up conclusions 5 and 6, leaving enough cresylic acid for good frothing. These conclusions yielded good reproducible results because of the ample frothing, which allowed similar amounts of froth to be raked off in any two tests. Conclusion 3 yielded a large standard deviation, not explainable by the above. It could be caused by an error in measurement of reagents, sampling, etc. It does not materially affect, however, the final conclusions.

Analysis of results at pH 10.5, Table IX, shows that none of the interactions are significant. Table IX then becomes Table XIV. The F test of Table XIV given in Table XV.

Table XIV. Main Effects at pH10.5

Main Effect	Degrees of Freedom (n-1)	Sum of Squares nS^2	Variances $\frac{nS^2}{n-1}$
G	1	294.12	294.12
X	1	46.31	46.31
C	1	0.97	0.97
F	1	164.48	164.48
Residual	11	223.74	20.34

Table XV. F Test of Table XIV

Main Effect	Main Effect Variance ÷ Residual Variance	5 Pct Level of Significance from Tables
G	14.5	4.8
X	2.3	4.8
C	Less than Residual	4.8
F	8.1	4.8

At a pH of 10.5, xanthate and cyanide variability are not significant. Only variation of grinding and frother affects the results. The conclusions are given in Table XVI.

These conclusions are physically reasonable. At a high pH, the chalcocite-pyrite middlings are depressed. Therefore, fine grinding is necessary to liberate the chalcocite from the depressed pyrite-chalcocite middlings. At this pH, cyanide has little effect on pyrite depression, since the high pH does an efficient job. The lesser amount of xanthate is sufficient to coat the copper minerals, and since the pyrite chalcocite middlings are well depressed, additional xanthate does not affect copper recovery materially. The recoveries were low with the smaller amounts of frother because cresylic acid at the high pH of 10.5 is fairly soluble,[9] and hence not much is available to cause frothing. Doubling the amount of frother increased the frothing and therefore the recovery of copper.

Validity of Conclusions

Conclusion 4 of Table XVI is made up of four tests, two variables of which were shown to be nonsignificant. These variables were cyanide and xanthate. If it could be shown experimentally that the results would have been substantially the same even if these variables were not varied, then the mathematical analysis would be confirmed. Accordingly four tests were run, using the conditions of test 32, Table I. The copper recoveries were: 84.65, 76.02, 78.01, and 83.39, or an average of 80.52 pct and standard deviation of 3.54 pct. This compares with conclusion 4 of Table XVI of 80.66 pct and a 3.92 pct standard deviation. Using the physical conditions of test 22, which conclusion 4 of Table XVI listed as similar to test 32, four repeat tests yielded the following recoveries: 77.02, 82.41, 77.95, 85.67, averaging 81.23 ± 3.38 pct standard deviation. Thus, test 22 is substantially the same as test 32. This was shown to be so by eight tests; factorial design arrived at the same conclusion, by rigorous mathematics, in two tests.

Table XVI. Conclusions at pH10.5
Xanthate and Cyanide Variability Nonsignificant

Conclusion	Significant Variables: Grind, Mesh	Significant Variables: Frother, Lb per Ton	Test No. in Table II	Average	Standard Deviation
1	65	0.05	2,4,1[illegible]	65.67	2.30
2	65	0.10	18,20,26,28	74.42	1.27
3	200	0.05	6,8,14,16	76.58	5.48
4	200	0.10	22,24,30,32	80.66	3.92

Table XVII. Grade of Concentrate Variances

Type Action	Source of Variance	Degrees of Freedom (n-1)	Sums of Squares nS^2	Variance $\frac{(nS^2)}{n-1}$
Main Effect	G = Grind	1	152.73	152.73
	X = Xanthate	1	2.52	2.52
	P = pH	1	153.60	153.60
	C = Sodium Cyanide	1	173.77	173.77
	F = Frother	1	0.03	0.03
First Order Interactions	G x X	1	20.15	20.15
	G x P	1	0.63	0.63
	G x C	1	9.08	9.08
	G x F	1	5.10	5.10
	X x P	1	22.24	22.24
	X x C	1	5.81	5.81
	X x F	1	1.06	1.06
	P x C	1	29.25	29.25
	P x F	1	63.83	63.83
	C x F	1	59.71	59.71
Second Order Interactions	G x X x P	1	4.27	4.27
	G x X x C	1	1.70	1.70
	G x X x F	1	6.88	6.88
	P x C x F	1	0.16	0.16
	P x C x X	1	0.05	0.05
	P x C x G	1	0.19	0.19
	P x X x F	1	18.51	18.51
	C x X x F	1	1.25	1.25
	C x F x G	1	3.35	3.35
	G x C x F	1	5.73	5.73
Third Order Interactions	G x P x C x F	1	12.32	12.32
	X x P x C x F	1	0.02	0.02
	G x X x P x F	1	0.25	0.25
	G x X x P x C	1	0.02	0.02
			3.97	3.97
Unknown Reactions	Residual	1	2.66	2.66
		Total	760.84	760.84

Analysis of Concentrate Grade

The test results of Table II are arranged similar to Table III, but the concentrate grade results are used. The various effects and interactions are obtained in a similar manner. In this case, the analysis was carried out for second and third order interactions as well. The results are shown in Table XVII.

Table XVIII. First Order F Test

Interaction	Variance	Interaction ÷ Residual	5 Pct Significant Level from Tables
G x X	20.15	8.3	4.49
G x P	0.63	Less than residual	4.49
G x C	9.08	2.4	4.49
G x F	5.10	1.3	4.49
X x P	22.24	5.8	4.49
X x C	5.81	1.5	4.49
X x F	1.06	Less than residual	4.49
P x C	29.25	7.7	4.49
P x F	63.83	16.6	4.49
C x F	59.71	15.6	4.49

Table XIX. Four Factor Experiment

Source of Variance	Variance P_1 (pH8.5)	Variance P_2 (pH10.5)
G	66.91	86.45
X	4.88	19.87
C	172.79	30.23
F	30.36	33.50
$G \times X$	2.92	21.50
$G \times C$	5.98	3.29
$G \times F$	10.83	0.00
$X \times C$	2.33	3.54
$X \times F$	5.30	14.19
$C \times F$	33.12	26.75
Residual (5 deg freedom)	3.80	2.68

None of the third order interactions are significant. The new residual is 3.21 with six deg freedom. When measured against this new residual, no second order reaction is significant. They are therefore pooled with the residual, and a new residual estimated: 3.83 with 16 deg of freedom.

Table XVIII shows that $G \times X$, $X \times P$, $P \times C$, $P \times F$ and $C \times F$ are significant interactions. The frother is involved in two interactions; xanthate in two; pH in two; cyanide in two. This five factor experiment must then be broken down into two four factor experiments, as explained previously. Table XIX gives the results of a test broken down at P_1 (pH 8.5) and P_2 levels (pH 10.5).

Table XX. Significant First Order Interactions

Source of Variance	Variance P_1 = pH8.5	Variance P_2 = pH10.5
G	66.91	86.45
X	4.88	19.87
C	172.79	30.23
F	30.36	33.50
$C \times F$	33.12	26.75
$G \times X$	not sig.	21.50
Residual	4.65 (10 deg freedom)	3.82 (9 deg freedom)

Analyzing the four factor experiments at pH = 8.5, only the $C \times F$ interaction is significant. At a pH of 10.5, the significant interactions are $C \times F$ and $G \times X$. Combining the nonsignificant first order interactions with the residual, gives the results shown in Table XX.

At a pH of 8.5 only the grinding variable is significant; the main effects xanthate, cyanide and frother are not significant. However, the interaction cyanide and frother is significant. Therefore, at a pH of 8.5 the following tests can be added, resulting in Table XXI.

At a pH of 10.5, none of the main effects are significant; only the two first order interactions, $C \times F$ and $G \times X$. Since these four variables are involved, and the other variable, P, is held constant at P_2 level, each test is significantly different from each other, at a pH of 10.5. No tests at a pH of 10.5 can be added and averaged. Each test represents a random sample of the results using the physical variables listed.

Test No. 22, Table II, yielding a 4.72 pct concentrate grade, was repeated four times, averaging 5.90 pct Cu with a standard deviation of 1.27 pct. Test No. 32, Table II, yielding a 12.66 pct concentrate grade, was repeated four times, averaging 11.68 pct Cu with a standard deviation of 0.97 pct. These two examples show that the conclusions arrived at by factorial design are valid because test No. 22 with a 5.90 ± 1.27 result is certainly different than test No. 32 with an 11.68 ± 0.97 result, and this conclusion with factorial design was arrived at with two tests. The experimental error is taken into consideration in the conclusion. In conventional testing, two tests yielding 4.72 pct and 12.66 pct respectively, may or may not represent differences caused by known variables. While in the laboratory unknown variables are usually kept to a minimum, particularly ore-variability and operator variability, this is almost impossible in plant tests.

Table XXI. Conclusions for Concentrate Grade at pH8.5

Conclusion	Significant Variables: Grind, Mesh	Significant Variables: Cyanide, Lb per Ton	Frother, Lb per Ton	Test No. Table II	Average	Standard Deviation
1	65	0.0	0.05	1,3	5.92	0.27
2	200	0.0	0.05	5,7	2.81	0.86
3	65	0.0	0.10	17,19	5.56	1.00
4	200	0.0	0.10	21,23	2.93	0.58
5	65	0.10	0.05	9,11	8.95	2.05
6	200	0.10	0.05	13,15	7.17	1.82
7	65	0.10	0.10	25,27	18.11	0.90
8	200	0.10	0.10	29,31	9.27	1.13

Acknowledgments

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Twisted Return Runs for Conveyor Belts

by J. W. Snavely

WITH all the advantages of handling bulk materials by means of belt conveyor also go some problems, one of the most persistent being that of cleaning. When sticky materials are being carried; the build-up of material on the return idler rolls results in difficulty of belt training. Much attention has been given to the problem of cleaning conveyor belts, and a great variety of cleaning devices have been developed. Even the best involve troublesome maintenance, and none can completely remove the fine particles imbedded in the belt cover, which cause rapid wear of the return idler rolls, and at the same time, of the belt cover as well.

One of the major rubber companies has been promoting a two-way belt system, in which the conveyor belt is given two successive 90° twists at each end to enable it to carry material in both directions simultaneously. For many years the flat belt transmission industry has installed quarter-turn and half-turn twists in countless numbers of instances.

While quarter-turn and half-turn twists in transmission belts is a familiar application, the 180° twist apparently has never been previously attempted with a conveyor belt. About a year ago, two officials of the National Iron Co. of Duluth, Lester and Lewis Erickson, proposed twisting the return run of a conveyor belt on an installation that they were designing for one of the major iron ore producers. Since then the soundness of the idea has been demonstrated, both in theory and by practical test, with the result that the installation of two conveyor belts involving the twisting of the return run is now under way.

These two installations are designed to have the return run of the conveyor belt twisted 180° as it leaves the snub pulley at the head drive. The clean underside of the belt is thus placed against the idlers on the return run as well as on the carrying run. Just before it enters the tail pulley, the belt will be twisted an additional 180°, restoring it to its normal position.

Because this twisting of the return run of a conveyor belt is a radical departure from accepted practice, an elaborate and extensive test was conducted early in 1950 to demonstrate that this twisting of the return run could be done successfully, also to establish application data for accomplishing this twisting, and to determine if any special equipment would be required. In studying this concept of twisting the return run of a conveyor belt, a number of problems need to be solved, primarily the ones brought about by deliberately introducing an unequal distribution of stress across the conveyor belt and controlling that maldistribution of stress, while confining it to the return run portion.

The tension conditions existing in the return run of a conveyor belt are clear to all designers. First, the return run carries the initial or slack side tension of the conveyor belt, the tension that must be supplied to the return run to provide proper frictional contact between the belt and the driving pulley so that the necessary power can be transmitted from the driving pulley to the belt without slippage. This slack side tension is supplied to the belt by means of takeups, either of the gravity type, which can be vertical or horizontal, or by means of the screw type.

With inclined or declined belt conveyors the slope tension also must be considered, which is the tension imposed by the weight of the belt hanging from the top pulley. This slope tension frequently can furnish part or even all of the initial tension required. The maximum value of the slope tension will be at the top pulley, and it decreases in direct proportion to the length.

In addition to the foregoing, it frequently is desirable to impose arbitrarily additional slack side tension to provide sufficient tension at the loading point at the tail, so that the belt will adequately support its load between the carrying idlers.

Design Conditions for Twisting

A number of design conditions exist, which must be satisfied successfully to accomplish the twisting of the return run without exceeding normal working limits in any portion of the conveyor belt. It is obvious that the belt edge, in its relation to the center of the belt, must stretch in making a twist, because as the twist is accomplished, the belt edge travels through a longer path than does the center of the belt.

It is further obvious that if the edge of the belt is stretched, a redistribution of stress in the belt is required to allow this edge stretching. Moreover, this stress will be unequal across the width of the belt, having a maximum value at the edges, with a minimum value at the center of the belt.

With correct initial tension in the return run of a conveyor belt, the existing slack side tension will be unequally distributed when a twist is introduced. A condition then exists in which the edge stresses,

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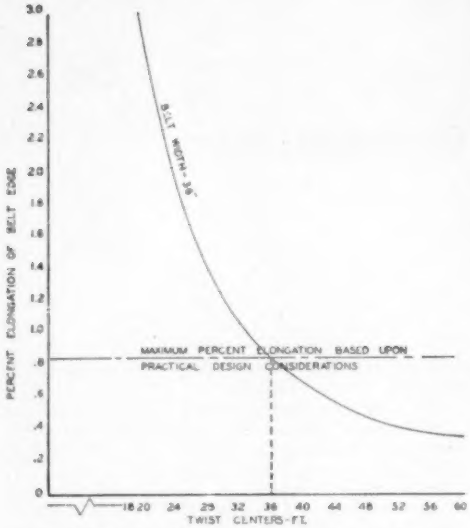

Fig. 1—Tentative minimum twist centers selection chart.

in pounds per inch ply, will be higher than the initial stresses normally uniform across the belt width, and the central stresses will be lower.

Further, when the unequal distribution of stress caused by twisting produces a minimum stress in the central portion of the belt, which is zero or less, the belt has a tendency to *cup* or collapse throughout the twist length. Consequently, to impart a stable condition to the belt in making the twist, tension sometimes must be added to the return run, over and above normal requirements.

Further study shows that the minimum initial tension required to elevate the minimum stress portion of the belt to zero and above is dependent upon the belt width, twist distance, belt elastic modulus, and the number of belt plies. It is necessary to know the elastic modulus of the belt, and it also is obvious that the belt width affects the design because of the longer path that the belt edge of the wider belts travels in negotiating the same twist. By the same token, the twist centers also directly affect the percentage of elongation of the belt edges in relation to the center.

Design Problems

As a matter of practical expediency, existing belt constructions must be used rather than attempt to set up specifications for a special belt construction. In working out the application of twisting the return run of a conveyor belt, it becomes necessary to determine several design components.

1—The twist centers must be established for a given belt width to hold the elongation of the belt edge with relation to the belt center within the working limits of belt construction used, while twisting will usually be carried out in the horizontal plane; doing so in the vertical plane is also practical and both conditions must be studied. From a practical standpoint it is desirable to accomplish the twist in a minimum distance.

2—It must be determined that the normal required initial tension obtained from standard design practices allows a positive stress in the central portion of the belt for the particular belt characteristics and twist centers selected to insure that the belt will be in a stable condition. If necessary, additional tension must be added to meet this requirement.

3—The belt edge stress must be checked to insure that it is below the working limits of the belt construction used.

To show clearly what must be done, appropriate formulae and sample computations used in working out a sample problem involving a 36-in. belt have been prepared. This is the same belt width used in the tests which will be described later. The following is a sample problem which will be used, assuming for illustrative purposes only, that one twist will be in a vertical plane and the other in a horizontal plane.

Stockpile belt to handle iron ore with a tripper on horizontal run. Material, iron ore, 150 lb per cu ft; Capacity, 1000 tons per hr; Belt speed, 400 ft per min; Centers, 600 ft 0 in. horizontal, 40 ft 0 in. vertical; Belt specifications, 36 in. 6 ply, 36 oz duck with ¼-in. and 1/16-in. covers, weighing approx. 12 lb per ft.

Normal Design Tensions, lb:

Effective tension,	et =	6110
Initial tension,	t =	2020
Total tension,	TT =	8130
Tension at foot,	T =	1800
Slack side tension at head,	T =	2020

Normally, the arrangement of the mechanical equipment for this example would indicate that twists in the horizontal plane would be employed at both the head and tail. However, for illustrative purposes only, a vertical twist will be assumed at the tail and horizontal twist at the head. For purposes of facilitating the illustrative calculations, the belt elastic modulus used will be 2170.

Twist Centers

The selection of twist centers must keep the amount of stretch imposed upon the belt edge in relation to the center, as measured by the percentage of elongation, within the ability of the belt to resist cupping or collapse. As will be shown, a workable rule of thumb for the initial selection of twist centers is to use a twist distance in feet equal to the belt width in inches. Arrived at independently, it confirms the practice that has been successfully used in the transmission belting industry for years.

Based upon these primary considerations, the tentative twist centers for the illustrative problem is selected from Fig. 1. The path transcribed by the belt edge is that of a helix and can be determined by the triangular method. The edge of the belt is the hypotenuse of the triangle; the longer leg, the length between the pulleys; and the shorter side, the circular arc distance transcribed by a point on the edge of the belt in twisting 180°.

The formula which can be used to determine the length of the belt edge for a particular belt width and twist centers is as follows:

Length of Belt Edge in Twisting 180°: Using the following symbols: W = width of belt, in., L = twist centers, in.

$$\text{Length of Belt Edge} = \sqrt{\left(\frac{\pi W}{2}\right)^2 + L^2}$$

From example: Belt width, 36 in.; Tentative twist centers from Fig. 1—36 ft 0 in.

$$\text{Length of Belt Edge} = \sqrt{\left(\frac{\pi 36}{2}\right)^2 + (36 \times 12)^2}$$

$$= 435.68 \text{ in.}$$

The percentage of belt edge elongation with respect to the center is shown in the following formula, clearly indicating that higher stresses will be present in the edge:

$$\frac{\sqrt{\left(\frac{\pi W}{2}\right)^2 + L^2} - L}{L} \times 100$$

$$= \frac{\sqrt{\left(\frac{\pi 36}{2}\right)^2 + (36 \times 12)^2} - (36 \times 12)}{36 \times 12} \times 100$$

$$= 0.852 \text{ pct}$$

Twist Tension

As previously outlined, the return belt must be in a stable condition throughout the twist length, requiring that the minimum stress in the central portion of the belt be zero or above. Therefore, it must be determined that the normal required initial tension obtained from standard design practices, together with the particular belt and twist length selected, permits or allows this zero or positive stress in the central portion of the belt throughout the twist. Theoretically, the minimum stress at the central portion of the belt can be zero, but again, a factor of safety is desirable to positively insure the desired stable condition. It is advisable therefore, that this minimum stress be equal to or greater than 0.2 pct times the elastic modulus of the belt. This value is an arbitrary one and conceivably could be varied. However, it is reasonable in magnitude as indicated by the test conducted.

It is convenient to divide the minimum required twist tension into two parts: 1—the tension required to elevate the minimum stress to zero, and 2—the tension required to elevate the minimum stress to insure a stable belt section. Fig. 2 illustrates point 1 and the following formula shows the derivation of point 2 together with the additional slack side tension to be added to the normal tension to arrive at the minimum required twist tension.

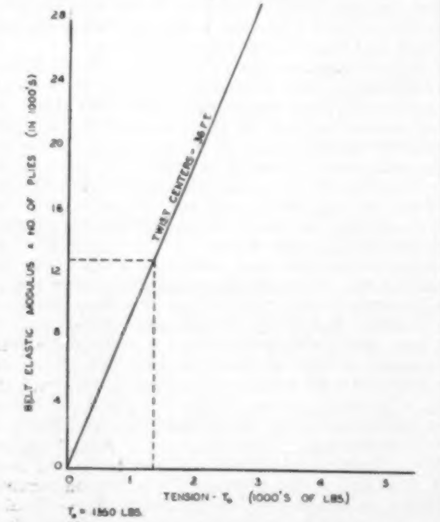

Fig. 2—Tension required to elevate minimum stress to zero.

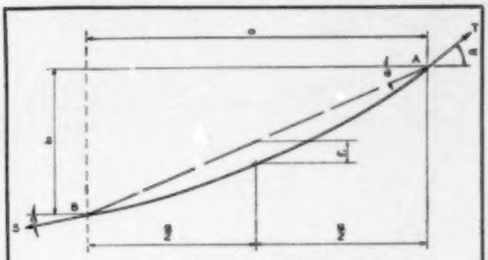

Fig 3—Theoretical sag.

Using the following symbols: a = Horizontal distance between points of belt supports, ft; b = Vertical distance between points of belt supports, ft; θ = Angle, which AB makes with the horizontal, referred to as angle of inclination of twist ($= \tan^{-1} \frac{b}{a}$); T = Twist tension at head which is equal to new T_s; α = Approach angle for centerline of belt at upper point of suspension-degrees; β = Approach angle for centerline of belt at lower point of suspension-degrees, f_1 = Theoretical sag, ft; $n_1 = \frac{f_1}{a}$ = sag ratio; w = wt of belt, lb per ft.

$$T = T_s + \Delta T$$
$$= 2020 + 488$$
$$= 2508 \text{ lb}$$

$$T = \tfrac{1}{2} wa \left[\frac{1}{16 n_1^2} + \frac{\text{sine } \theta}{2 n_1} + 1\right]^{1/2}$$

$$2508 = \tfrac{1}{2} (12) \quad (36) \left[\frac{1}{16\left(\frac{f_1}{36}\right)^2} + \frac{\text{sine } 8^\circ}{2\left(\frac{f_1}{36}\right)} + 1\right]^{1/2}$$

$$f_1 = 0.791 \text{ ft} = 0 \text{ ft} - 9\tfrac{1}{2} \text{ in.}$$

Using the following symbols: T = Minimum required twist tension, lb, T_m = Tension required to elevate minimum stress to insure a stable belt section throughout the twist, lb, T_o = Tension required to elevate minimum stress to zero, lb (obtained from Fig. 2), ΔT = Additional slack side tension to be added, lb, t_m = Minimum stress to insure a stable belt section, lb per in. ply, W = Width of belt, in., P = Number of plies, T_s = Tension at foot, lb, t_m = Belt elastic modulus × 0.2 pct.

$$T_m = t_m WP$$
$$= 0.2 \text{ pct } (2170)\ (36)\ (6)$$
$$= 938 \text{ lb}$$

$$T = T_m + T_o$$
$$= 938 + 1350$$
$$= 2288 \text{ lb which becomes the new } T_s$$

$$\Delta T = T - T_s$$
$$= 2288 - 1800$$
$$= 488 \text{ lb}$$

It has been shown that the belt edge stress will be elevated above the normal initial belt stresses because of the twist. It is necessary therefore to check the belt edge stress to insure that it is below the working limits of the belt used as shown by the fol-

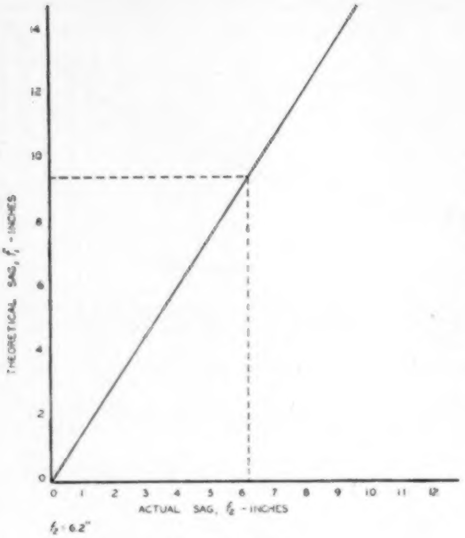

Fig. 4—Actual sag.

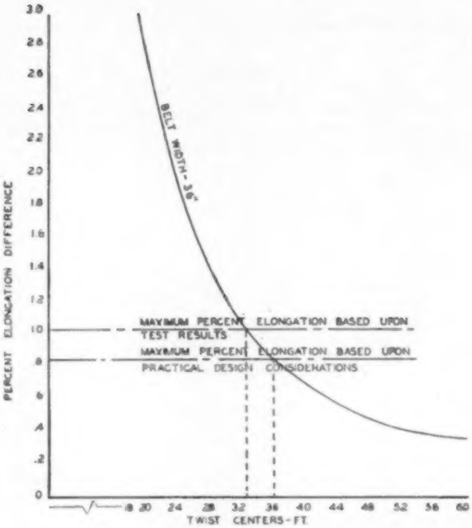

Fig. 5—Tentative minimum twist centers selection chart.

lowing formula. Examination reveals that the belt elastic modulus becomes a very important consideration in the selection of belts to satisfy this requirement.

Belt Edge Stress (Vertical Twist), Using the following symbols: T = New T_b, twist tension, lb, P = Number of belt plies, W = Width of belt, in., t_e = Maximum stress at edge of belt, lb per in. ply, t_m = Minimum stress at center of belt, lb per in. ply.

$$T = PW\, t_m + \frac{PW}{3}\,(t_e - t_m)$$

$$2288 = 6\ (36)\ (4.34) + \frac{6\ (36)}{3}\,(t_e - 4.34)$$

$$t_e = 23.12 \text{ lb per in. ply}$$

Horizontal Twists

It can be appreciated that the foregoing is applicable to a vertical twist because the distribution of stress is symmetrical about the centerline; that is, the stresses of both edges will be equal. However, twists completed in a horizontal plane are acted upon by the external force of gravity, requiring further analysis.

A conveyor belt is flexible to a certain degree, and for that reason the curve which the belt transcribes between two pulleys when suspended in a horizontal plane approaches that of a catenary in form. The end result is that the belt edges stretch unequally, causing a difference in belt edge stresses. The solution of the highly complex exponential form of catenary equations is extremely difficult, and since the results for low sag ratios, where the sag is less than 10 pct of the span, are for all practical purposes the same, the solution used follows the parabolic form of suspension.

However, the belt does possess some longitudinal rigidity, because of its cross-section in the twist, the magnitude of which is dependent upon its width and construction. The amount of actual sag of the belt between the twist pulleys therefore will be appreciably less than the theoretical amount of sag if the belt were treated as a completely flexible member.

It should be clear that the amount of actual sag determines the maldistribution of stress across the width of the conveyor belt. As mentioned above, the determination of the theoretical amount of sag is that of a parabolic function, and Fig. 3 shows the formula and how it is applied to the example used.

The amount of difference between theoretical and actual sag is largely a function of the moment of inertia of the belt about its horizontal axis at any specific point of its rotation throughout the twist, together with the belt characteristics. For the 36-in. belt and the construction used in the test, the relationship of the actual sag to the theoretical sag was 0.652, and Fig. 4 shows this relationship.

In the twisting of the return belt in the horizontal plane, one of the critical problems is to limit the stress in the high tension edge of the belt. After the sag calculations are made, it is necessary to check that the stress in the high tension edge be below the recommended maximum rating of the belt. As the maldistribution is dependent upon the actual amount of sag, the effect of this sag must be considered. The mathematics involved become somewhat complicated but are best worked out knowing that the stress distribution curve across the width of the belt follows that of a parabola. The formula used, and its application to the example are illustrated below:

Critical Belt Edge Stress (Horizontal Twist): Using the following symbols: X = Distance from low tension edge to minimum stress, in., Y = Distance from high tension edge to minimum stress, in., t_m = Minimum stress, lb per in. ply, t_{eH} = Stress at high tension edge, lb per in. ply, t_{eL} = Stress at low tension edge, lb per in. ply, f_2 = Actual sag, in., W = Width of belt, in., T = Twist tension at head, lb, E = Belt elastic modulus, T_o = Tension required to elevate minimum stress to zero, lb, T_m = Tension required to elevate minimum stress across width of belt, lb.

$$Y = \frac{W}{2} + f_s \qquad X = \frac{W}{2} - f_s$$

$$= \frac{36}{2} + 6.2 \qquad = \frac{36}{2} - 6.2$$

$$= 24.2 \text{ in.} \qquad 11.8 \text{ in.}$$

$$T_m = T - T_e \qquad t_m = \frac{T_m}{WP}$$

$$= 2508 - 1350 \qquad = \frac{1158}{36(6)}$$

$$= 1158 \text{ lb} \qquad = 5.36 \text{ lb per in. ply}$$

$$T_e = \frac{P(X^2 + Y^2)\ (t_{eH} - t_m)}{3\,Y}$$

$$1350 = \frac{6(11.8^2 + 24.2^2)\ (t_{eH} - 5.36)}{3(24.2)}$$

$$t_{eH} = 27.95 \text{ lb per in. ply}$$

For a 36-in. belt, the test results showed that the minimum twist centers in which the belt could be twisted, analyzed from the standpoint of tendency to collapse because of the uneven distribution of stress, was 1.04 pct elongation of the edge with respect to the center. This was correlated to theoretical minimum twist centers and substantiated by test results.

Sound engineering practice dictates the use of a factor of safety in design work such as this, and a commonly applied factor is 80 pct of theoretical value. Eighty pct of the 1.04 pct of elongation difference gives a working value of 0.832 pct permissible elongation difference. For convenience, this is slightly increased to 0.852 pct, because at that figure, the twist centers in feet are exactly equal to the belt width in inches.

Fig. 5 has been prepared to show the percentage of elongation curve for the 36-in. belt, together with the limiting design percent of 0.852. This figure actually is an overlay of Fig. 1 with the minimum percent elongation of 1.04 sketched in for comparative purposes.

In analyzing what takes place during the twisting of the return belt, the term approach angle must be used. In leaving and entering the twist pulley, the conveyor belt does not contact the twist pulley in the same plane. Going into the twist, the belt has a substantially uniform distribution of stress across its width, and the approach angles of the edges vary equally with the approach angle of the center of the belt. This variation is largely independent of the centers of the twist and the amount of sag and tension but can be more closely correlated to a function of width and transverse rigidity of the belt.

In leaving the twist, the belt has an uneven distribution of stress, imposed by the twisting, and the approach angles of the edges vary unequally from the approach angle at the center of the belt, with one edge of the belt having a greater variation than the other. In general, however, the pattern of a helical generation is followed.

Fig. 6 illustrates what is meant by these approach angles.

It is essential that there be sufficient contact between the return belt and the twist pulleys to permit these twist pulleys to influence and train the belt. This amount of contact is commonly referred to as angle of wrap, and in accomplishing the twisting of the return belt, this is most critical on the leaving twist pulley, where a minimum of 5° of wrap is required at the minimum stressed edge.

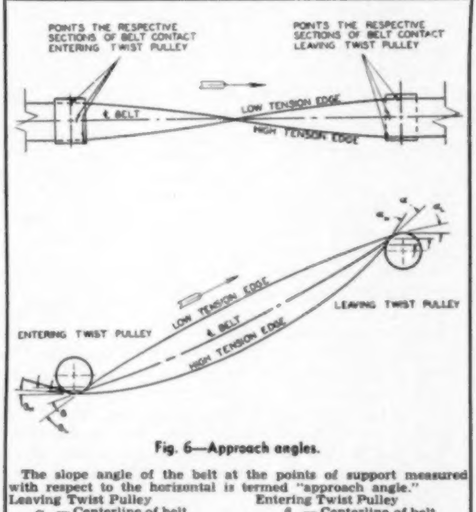

Fig. 6—Approach angles.

The slope angle of the belt at the points of support measured with respect to the horizontal is termed "approach angle."

Leaving Twist Pulley	Entering Twist Pulley
α — Centerline of belt	β — Centerline of belt
α_L — Low tension edge	β_L — Low tension edge
α_H — High tension edge	β_H — High tension edge

Using the following symbols:
f = Theoretical sag, ft
a = Horizontal distance between points of belt supports, ft
b = Vertical distance between points of belt supports, ft

$$\tan\alpha = \frac{b + 4f_1}{a} = \frac{5.05 + 4(0.791)}{36} = 0.228 \qquad \alpha = 12° - 50'$$

$$\tan\beta = \frac{b - 4f_1}{a} = \frac{5.05 - 4(0.791)}{36} = 0.0524 \qquad \beta = 3° - 6'$$

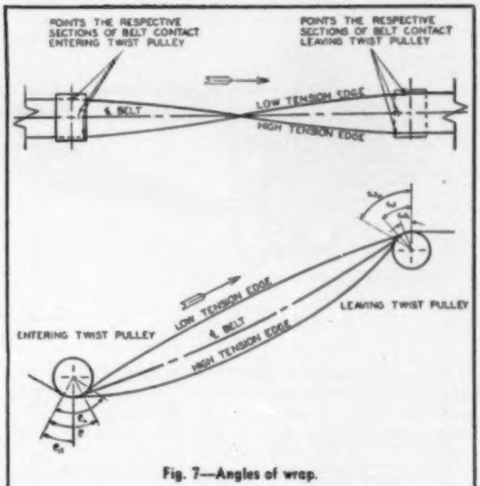

Fig. 7—Angles of wrap.

The angular measurement of belt contact on the twist pulleys is termed "angle of wrap."

Leaving Twist Pulley —
ω — Centerline of belt
ω_L — Low tension edge
ω_H — High tension edge

Entering Twist Pulley —
θ — Centerline of belt
θ_L — Low tension edge
θ_H — High tension edge

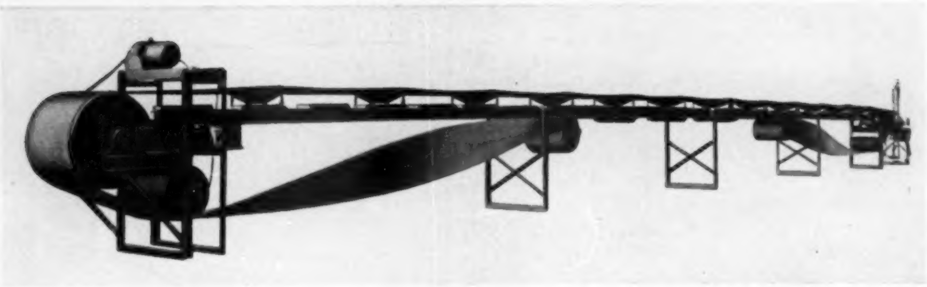

Fig. 8—Test conveyor.

For a given twist centers and belt weight, the angle of wrap is dependent upon the approach angles and the relative position of the belt travel with relation to the pulleys. Fig. 7 illustrates these angles of wrap.

Largely based upon analysis of the test results, the following comments on the various equipment items seem appropriate. Generally speaking, no special equipment is going to be required to twist successfully the return run of a conveyor belt.

It should be possible to accomplish successfully the twisting of the return run of conveyor belts using standard fabric constructions of belting and some low modulus cord constructions as well. The final determination of the twist centers necessarily will need to take into consideration the construction of the belting used, and it is essential that the elastic modulus of the belt, within reasonable limits, be obtained from the belting supplier.

Until a number of successful applications have confirmed the design procedure, it will be essential to have the belting supplier also make a simple static twist test with a sample piece of the actual belt to be supplied. This is not particularly difficult or expensive to set up, and will serve as a final important check on the design work that has been done.

Mechanical Equipment

The pulleys used in accomplishing the twist normally would be referred to as bend pulleys, but because of their particular purpose, they have been and will be referred to as twist pulleys. Because of the uneven distribution of stress across the width of the belt in accomplishing the twist, flat-faced pulleys within normal tolerances of manufacture are necessary. Crowned pulleys would only add to the amount of adjustment necessary in belt training. Also, the pulleys should be slightly wider than normally used, about 4 in. wider than the belt up to widths of 36 in., and 6 in. wider than the belt for widths beyond 36 in. This additional width to the pulley face is merely to safeguard the edges of the belt should alignment difficulty develop during initial run-in period.

It is the leaving pulley of each twist that is the critical one, and the only pulley on which adjustment need be made. Such adjustment can be made entirely in the horizontal plane, and based upon the experience of the test, it is not anticipated that such adjustment should ever need to exceed 3 in. Small adjustments of ⅛ in. to ¼ in. produced a considerable belt training effect. Therefore, it would be recommended that an adjustable type of base plate, equipped with adjusting screw and locknuts be provided, and antifriction, self-aligning bearings used for the twist pulley mountings.

Any reasonable arrangement of pulleys can be used to complete the twist in satisfying the requirements of a specific installation problem, provided the necessary design considerations are carried out. It is emphasized that a twist pulley with adjustment facilities always be provided preceding the tail pulley, for central loading on the belt.

Idlers: Conventional idler equipment will be completely adequate, and there would seem to be no need for the use of training return idlers beyond normal requirements because of the very effective training influence of the leaving twist pulleys. The need for training idlers on the return run of the conveyor belt should be sharply decreased, because there will be no build-up of material on the return rolls tending to deflect the belt to one side or the other. The need for expensive special constructions of return idlers also disappears, and the conventional solid roll return idler can be used with complete assurance, with a much longer life expectancy than ever heretofore possible.

Takeups: It is obvious that very accurate control of tensions must be maintained at all times, and it therefore becomes essential that an automatic type of takeup be used to insure the proper amount of slack side tension at all times and under all conditions. This is best accomplished with the gravity type of takeup, either horizontal or vertical, and it further becomes important to see that the takeup is

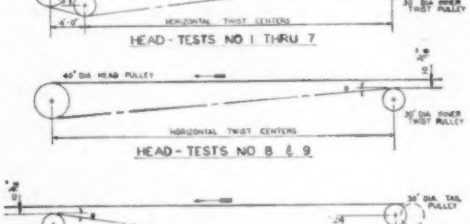

Fig. 9—Schematic arrangement of test conveyor.

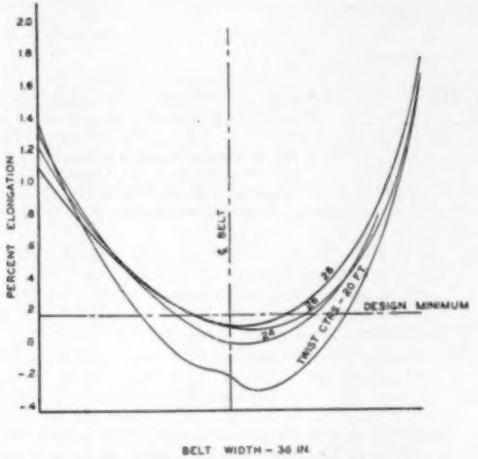

Fig. 10—Longitudinal elongation across width of belt, test results for short twist centers.

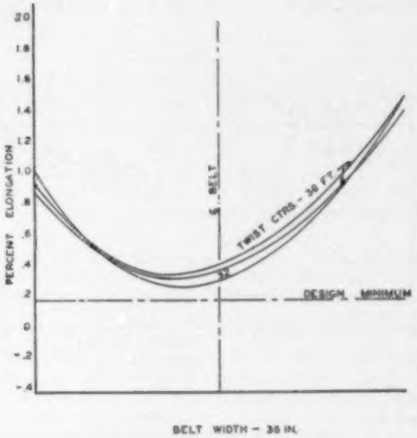

Fig. 11—Longitudinal elongation across width of belt, test results for more desirable twist centers.

free and operative at all times. It is desirable to have the takeup location where it is most sensitive, which is normally following the head pulley.

Drives and Controls: The use of customary drives and controls is completely satisfactory. A greater degree of control of accelerating tension becomes highly desirable to limit any surges of tension that might be sent through the conveyor belt, which could occasionally cause momentary fleeting of the belt on the twist pulleys.

Operating Techniques

A number of operating techniques became apparent from the test as it was conducted. The most important concerns the manner of introducing the twist in the belt. It is essential that the twist introduced into the return run of the conveyor belt be continued in the same direction so as to accomplish a total of 360°. If the belt is twisted 180° and then returned 180°, the same belt edge is being kept at maximum tension, which will have two results. First, the belt will be much more difficult to train, since it would appear that there is some buildup of residual stress that passes beyond the twist pulleys themselves. In addition, and more serious, over a period of time the belt would take permanent stretch to a greater degree along the continuous high tension edge than it would on the other edge, and successful belt training would then become virtually an impossibility.

It is also necessary for belts employing a twisted return run that only vulcanized splices be used. Not only is the mechanical splice, regardless of the type used, not as flexible as the belt itself, but it likewise does not distribute the stress across the belt in the normal pattern. Moreover, the mechanical splice will not permit the use of maximum tension ratings in applying the conveyor belt, and they are also particularly weak at the belt edge, which for twisting is the location at which they should be the strongest.

As to belt training, it is the leaving pulley that trains the belt, and the belt tends to run off on the low tension edge. The adjustment to train the belt can be made by moving the pulleys in a horizontal plane, moving the low tension edge of the pulley forward in the direction of belt travel. The belt is quite sensitive to such pulley adjustment, and a very small amount of movement immediately imposes the required training effect.

It would appear that speed has little influence on the belt operation, as the increase or decrease of

Fig. 12 — Stress-strain characteristic curves of conveyor belt fabrics.

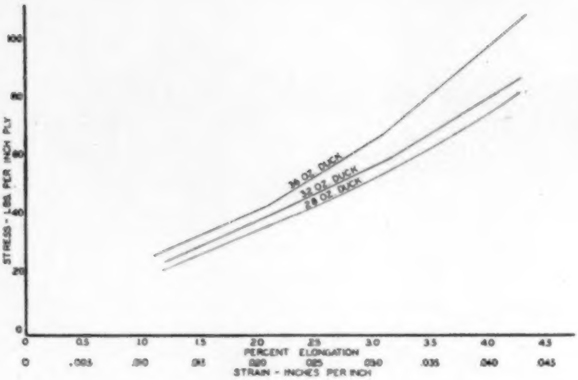

Fig. 13 — Stress-strain characteristic curves of conveyor belt fabrics, typical test data furnished by conveyor belt manufacturer B.

speed in the test conducted imposed no appreciable effect.

Acceleration control does become relatively important, as across-the-line starting places a surge of tension in the system. When the takeup is located so that it cannot react immediately to maintain a constant tension, there obviously would be a momentary decrease in the tension in the twist following the head pulley, and a decrease in slack side tension would tend to cause the return run to momentarily fleet on the twist pulley, being corrected when the conveyor is up to speed and the takeup counterweight has reacted.

To minimize this action, it is recommended that starting controls be used to limit the accelerating tension to approximately 135 pct of normal operating tension, and that the takeup be placed at the point of most sensitive reaction.

Tests and Results

The test that was conducted, and its confirmation of the mathematical basis of design is of considerable interest. The test program was a cooperative development project carried on by the National Iron Co. of Duluth, Minn., the Mining Department of Cleveland-Cliffs Iron Co. of Ishpeming, Mich., the Flat Belting Field Engineering & Development Div. of the B. F. Goodrich Co. of Akron, Ohio, and the Conveyor and Process Equipment Div. of the Chain Belt Co. The tests were performed May 10 to May 13, 1950, at the National Iron Co.

Fig. 8 shows the test conveyor setup which clearly shows the twisting at both the head and tail. Fig. 9 is a presentation of the schematic arrangement of the types of twisting employed in the conducting of the tests in question.

The following are the specifications of the test conveyor: *Belt*—Goodrich Longlife, 36 in., 5 ply, 32 oz duck with ⅛ x 1/16 in. covers, 2500 to 3000 psi tensile, 16 to 19 lb friction, belt weight 8 lb per ft (actual weight). *Belt speed*—271 and 526 ft per min. *Conveyor centers*—130 ft 1½ in. horizontal. *Takeup*—horizontal gravity takeup on tail pulley, counterweight weighed and variable. *Idlers*—REX style No. 32 troughing idlers with 5 in. diam rolls, inclined 2° in direction of belt travel at 10 ft spacing, REX style No. 55RC—return idlers at 10 ft spacing. *Drive*—Gearmotor, Westinghouse, 10 hp class I, 220/440 v ac motor, output rpm 84, 26/13 amp, ratio 20.8-1, 1750 rpm, unit No. CJA 324, 60 cycle; chain drive, No. 100 REX roller drive chain; 18 T, 7.198 in. P.D. driver for 271 ft per min; 70 T, 27.861 in. P.D.

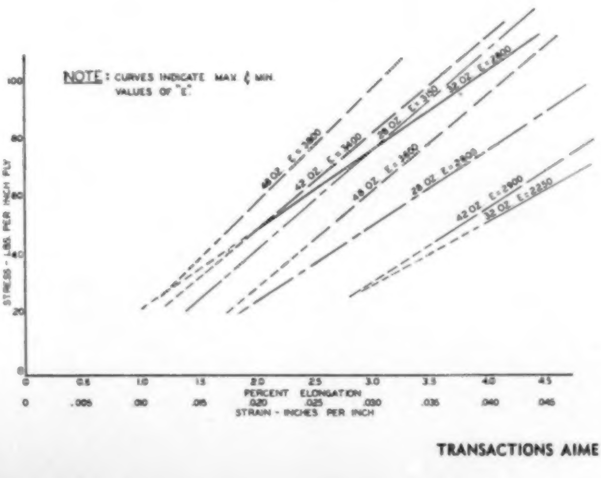

Fig. 14 — Stress-strain characteristic curves of conveyor belt fabrics, representative test data furnished by conveyor belt manufacturer A.

Table I. Test Data

Test No.	Twist	Length of Twist	Inclination of Twist	Twist Tension: Lb	Twist Tension: Lb, In. Ply	Sag, In.	Twist Pulley	Approach Angles, Deg: High Tension Edge	Approach Angles, Deg: Center	Approach Angles, Deg: Low Tension Edge	Angles of Wrap, Deg: High Tension Edge	Angles of Wrap, Deg: Center	Angles of Wrap, Deg: Low Tension Edge
7	Head	32 ft, 3¾ in.	7°-58'	1800	10	4 3/16	Outer	— 3	5	13	19	28	33
							Inner	17	12½	2	19	15	5
8	Head	36 ft, 1 11/16 in.	5°-05'	1800	10	6 1/16	Head	— 6	0	8	174	180	188
							Inner	14	8	1	16	12	3
9	Head	36 ft, 1 11/16 in.	5'-05'	1800	10	5⅝	Head	— 6	0	7	174	180	187
							Inner	14	10	1	13	13	3

driven; 35 T, 13.945 in. P.D. driver for 526 ft per min. *Pulleys*, head, 48 in. diam x 40 in. face with ½ in. thick vulcanized chevron type rubber lagging; tail, 30 in. diam x 40 in. face; head outside twist pulley, 30 in. diam x 40 in. face; head inside twist pulley, 30 in. diam x 40 in. face; tail outside twist pulley, 24 in. diam x 40 in. face; tail inside twist pulley, 30 in. diam x 40 in. face.

All of the pulleys are of the solid welded steel construction with straight face with the exception of the tail outside twist pulley, which could be classified as crowned, or an irregular flat face pulley.

After the conveyor was set up, the procedure following was used: The belt was painted with a rubber base Kemtone paint in three sections 6 ft apart. These were then scribed with longitudinal lines 33 in. long at 3-in. intervals transversely across the belt. Tension had been released to insure correct length under no stress. The tension was then reapplied and the elongation of the measured sections were tabulated. Readings were taken in the following positions: 1—Between head and outer twist pulley, 2—Three positions through head twist, 3—One position between inner head twist pulley and first return idler, 4—One position between the last idler and inner tail twist pulley, 5—Three positions through tail twist, 6—Between outer tail twist pulley and tail pulley, 7—One position immediately ahead of tail pulley on carrying side, 8—One position immediately behind head pulley on carrying side.

In addition to the strain readings mentioned, additional pertinent data taken consisted of recording the slack side tension, twist centers, slope of twist, approach of belt to the twist pulleys, actual sag, cupping, and angle of wrap on the twist pulleys.

Fig. 10 gives the results of four of the tests in which the twist was introduced through centers that were too short. It is to be noted that in every case the low points of the curves fall below the minimum recommended design figure of 0.2 pct elongation at the minimum stressed portion of the belt, and that two of the curves drop below zero.

Fig. 11 shows a similar presentation of the readings of three of the tests which were conducted at satisfactory twist distances, and it will be noted that in these instances, the low points of the curves are all above the design minimum. Study also reveals how closely the variables involved in the design of twisted belts are interrelated.

The remaining pertinent data for the three tests shown at satisfactory twist distances are given in Table I.

From the previous discussion, it is evident that the characteristics of the belt construction used are an important factor. However, the determination of these characteristics, in other words, the elastic modulus for the belt construction, is not always readily possible to establish to a precise degree. Because of the construction of conveyor belts, and the nature of the materials used, there is bound to be appreciable variation which cannot be avoided. The problem is clearly evident from a comparison of Figs. 12-14 which show the elastic modulus for several belt constructions from three different sources. It is therefore obvious, and it must be emphasized, that the approval of the belting supplier must be obtained to achieve successful belt performance.

Advantages

It must be immediately apparent that the basic advantage is the placing of the clean side of the belt on the return idlers. This will immediately provide freedom from many operating annoyances and specifically will provide such advantages as:

1—No build-up of material on the return idler rolls. Sticky materials and wet materials in freezing temperatures can be handled readily. 2—Much longer return idler life. 3—Increases belt cover life appreciably. 4—Elimination of all dribble under the conveyor, except underneath the twists. 5—Elimination of the need for deck plates except at the head and tail sections of the conveyor.

Disadvantages

There naturally are going to be some inherent disadvantages, although it would appear that these are heavily outweighed. Some of the disadvantages would seem to be:

1—The distance required for twisting makes it impractical for short conveyors. 2—It may be necessary to use slightly heavier back covers on the belts. 3—Additional pulleys are required. 4—Some sacrifice of working tension in the belt. 5—If the mechanical arrangement of pulleys is such that some pulleys are in contact with the dirty side of the belt, cleaning these pulleys may be necessary.

Conclusions

On paper, and from the test, the twisting of the return run of a conveyor belt would appear to have great promise of being an important step forward in the art of belt conveying. It must be remembered however, that the test conducted was of very short duration, and solely with an empty belt.

Good judgment dictates reserving final evaluation until its soundness has been demonstrated by actual and prolonged field operation. During the ensuing months this field experience will be gained, and it is confidently expected that a report a year from now will largely confirm the present promise.

Acknowledgments

Complete credit for the mathematical analysis and derivations presented is hereby given to R. A. Zettel of Chain Belt Co. Acknowledgment is also made of the assent by the members of the development project test for the presentation of this paper.

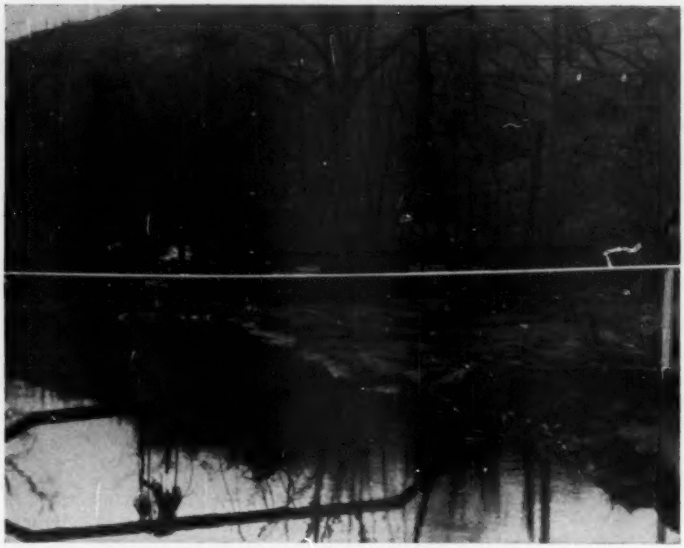

Fig. 1—Low spot in an acid polluted stream.

The Formation of Acid Mine Drainage

by Kenneth L. Temple and Arthur R. Colmer

ACID coal mine drainage presents a peculiarly difficult problem for two principal reasons. First is the fact that the amount of acid water discharged from active and abandoned mines constantly increases as coal fields are developed and thereby creates a steadily worsening nuisance of rapidly growing importance to the general public as well as to the coal industry. Second is the even more significant fact that there is no method yet known that will appreciably alleviate the acid water problem except at totally prohibitive cost to the coal operators. The problem is thus seen to be an urgent one. With a growing need for good quality water and a growing public demand for abatement of stream pollution, a large portion of the soft coal industry finds itself in the position of year by year dumping more and more acid into the streams and waterways and having no way in sight of improving the situation or even of keeping it from getting worse.

While some studies have been made, it is only in the last few years that the problem has been seriously considered, chiefly by the group of workers under the direction of S. A. Braley at Mellon Institute and by the Bituminous Coal Research, Inc. fellowship at West Virginia University. Hinkle and Koehler[1] reported the results of the earlier West Virginia work. This work pointed to certain biological aspects which had not been noticed previously. These may be summarized as two separate findings. One was that sulphur oxidizing bacteria were present in acid mine water and that their possible role in contributing to the acidity should be investigated. The other was

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that the oxidation of ferrous iron in acid mine water to produce the characteristic red color took place only under conditions which showed that the process was a biological one and not a simple atmospheric oxidation. Since that time the biological aspects of acid mine water have been studied more intensively, and some of the results obtained constitute the subject of this report.

All acid mine water is formed by the passage of ordinary ground water through a coal mine which may be either active, inactive, or even long abandoned. However a similar product often results from leaching of gob piles. The original ground water is never acid but may be neutral or slightly alkaline. On passing through an acid forming mine the water becomes acid in reaction. It develops a high titratable acidity when measured against a strong base such as caustic. It is found to contain a high content of inorganic salts, particularly ferrous iron and sulphate. Calcium, magnesium, and aluminum levels are also increased. The metallic ions and sulphate ions balance each other so that it may be said that there is no free sulphuric acid, in spite of the actual high acidity. No acid is formed in a coal seam which has not been opened up by mining. All acid water sooner or later undergoes a change in which the ferrous iron Fe^{++} is oxidized to ferric iron Fe^{+++}; the ferric content is responsible for the brown or red color. The quantitative acidity and hardness of acid waters covers all gradations from unaltered ground water to a highly acid water having a pH of 2 to 3, a total titratable acidity of several thousand ppm and an iron content of perhaps 2000 ppm Fe.

Ferrous Iron Oxidation

With regard to the discovery previously mentioned, that the ferrous to ferric change is biological, it may be said that this has been fully substantiated. The reaction is brought about by a bacterium, not hitherto known, which causes the oxidation to proceed in spite of the fact that ferrous iron is stable to atmospheric oxidation at the low pH levels encountered. While much work has been done on this phase of the problem[2] it is not of direct concern to the subject of this paper and will not be further discussed.

Sulphur Oxidizing Bacteria

The sulphur oxidizing bacterium discovered by Colmer and Hinkle[3] turned out to be *Thiobacillus thiooxidans*, a bacterium that has been known and studied quite intensively for many years. For the purposes of this problem the physiological characteristics of the organism are of prime importance. It belongs to a group of bacteria which live upon inorganic matter entirely, and get all of their nutrient requirements therefrom. They grow and reproduce by turning inorganic matter into organic cell material. One of the requirements of all forms of life, just as of any engine, is a fuel, a source of energy. *Thiobacillus thiooxidans* uses inorganic sulphur, the element and its compounds, as fuel. It oxidizes sulphur, or thiosulphate or tetrathionate into sulphate whereby it manufactures sulphuric acid. The question whether disulphides are similarly oxidized has been investigated before but not satisfactorily answered. The most that may be said is that soils or composts containing pyrite develop a large population of *T. thiooxidans* as the acidity develops.

A survey of acid mine waters of the Morgantown area from different coal seams and including a few samples from other parts of the state and from Pennsylvania showed that *T. thiooxidans* is present in all acid mine waters examined and may reasonably be supposed to be in all acid mine waters. The organism is not found in other streams and has never been encountered in nature except where oxidizable sulphur compounds were present. Since it is not believed to grow in any other way than by oxidizing sulphuritic material, it is a logical assumption that *T. thiooxidans* when found in acid water is actively contributing to the acidity of that water.

Upper Freeport Seam

During an investigation of a mine on the Upper Freeport seam more specific evidence of bacterial association with acid formation came to light. This particular mine was in a very early development stage and was not near any other mines. It contained one area that was very wet, most of the mine being dry. The wet area proved to be acid forming and an attempt was made to study the exact location of acid formation. This represented the first opportunity to see acid formation occurring inasmuch as older mines usually have acid formation largely restricted to worked out and inaccessible areas. By a simple pH test using an indicator solution on white filter paper applied to the damp surface of the wall, it was possible to find the source of acid in this mine.

Only a very small portion of the wet area was acid forming. The entire section was characterized by an unusually bad top for the seam, a slickensided shale of loose structure which could not be held in place. In consequence, the height of the entry at that point was almost 7 ft, although the coal measured only 40 in. Work on this entry was stopped, and eventually connection was established by driving back toward it, using both short posts and roof bolts which held the roof in place. The exposed roof strata consist of a soft shale, about a foot of rider coal (much thicker here than is usual) and more shale. Tests for pH indicated that most of the water was essentially neutral and similar to ordinary ground water. This applies to both roof drips and the water on the ribs. In a very few localized spots the water was highly acid. These spots were present in the shale and in the rider coal but not in the Freeport coal proper. These spots when in the rider coal could also be distinguished visually by the deposits of

Fig. 2—Typical black iron pipe corroded from inside. These pipes are replaced approximately once a week.

yellow copiapite, a basic hydrated iron sulphate which is an oxidation product of pyrite and is acidic in reaction but has a strong superficial resemblance to elemental sulphur. Acid spots in shale could not be detected visually. There were numerous small pyritic concretions resembling miniature sulphur balls. These did not test acid, and their distribution bore no relation to the location of acid spots. *T. thiooxidans* was found in the rider coal and shales wherever there was acid. It was absent from adjacent neutral areas. Only the surface was acid. When the surface was scraped away, the underlying material was neutral and contained no sulphur oxidizing bacteria. In a few days this newly exposed surface became acid and contained a vigorous population of *T. thiooxidans*.

These observations on the Upper Freeport seam lead one to conclude that the readily oxidizable sulphuritic material is in the roof strata of this seam and is of exceedingly local distribution, accounting for the fact that Upper Freeport water is only moderately acid when the discharge from an entire mine or area is considered. Furthermore the oxidation of the rapidly oxidizable sulphuritic material, whatever its nature may be, is intimately associated with the development of the sulphuric acid manufacturing bacterium *T. thiooxidans*. In addition it is evident that the acid spots contained a sulphuritic material much more readily susceptible to oxidation than the prominent and visibly apparent pyritic concretions. It is of interest in this connection that so far all laboratory experiments with *T. thiooxidans* have been conducted with the larger pyritic concretions and not with the acid forming shales. There is no question but that acid formation in this mine could be stopped if the surfaces of the roof and walls were coated with some substance sealing them against air. This might prove difficult to accomplish.

Pittsburgh Seam

The Pittsburgh coal in the vicinity of Morgantown is of two distinct types as regards acid water production. The two types are separated geographically by the Monongahela River. This distinction coincides with a difference in the roof structure. On the western side of the Monongahela River the Pittsburgh coal has a characteristic roof comprised successively of shale, coal, shale, coal, dense hard shale, and soft shale up to limestone. This is a general picture and, depending on the particular location, these layers may sometimes be subdivided further. At one point in an acid producing mine these strata are 25 to 30 ft thick between the coal and limestone. Above the Pittsburgh the Sewickley seam and sometimes the Redstone seam is present. On the eastern side of the Monongahela River between the Cheat River and Decker's Creek the roof strata consist of a shale and sandstone, with thin cover and no limestone or coal overhead. The shale, moreover, is exceedingly variable and may be entirely absent, or have a thin layer of sandstone between it and the coal. The sandstone is of two types, one yellow, and the other light gray to white.

The sources of acid in the Pittsburgh coal are several in number. These include pyritic concretions or sulphur balls, pyritic binders and top and bottom coals which have a higher sulphur content than the mined coal. In addition most of the strata up to the limestone or sandstone, as the case may be, are acid forming. It is suggested that the quantitative difference in acidity of the two parts of the Pittsburgh seam described are caused by the differences in roof structure rather than to the coal. All of the strata above the coal to the west of the Monongahela are highly acid forming. This is shown by their high content of oxidized sulphur in old areas. Table I gives the result of an analysis made on a series of strata in a mine which has produced much acid water but in a section which is dry at the present time. Ordinarily no appreciable sulphate sulphur is found in fresh samples. In this case where an old fall and the absence of rock dusting had allowed oxidation to proceed, the sulphate sulphur varied from 0.37 to 2.72 pct in the exposed strata. The lowest value of 0.37 pct is for the dense shale known as black rock which commonly forms the anchor for roof bolts in this seam. Individual spots could be noticed where the acid salts, principally Starkeyite (ferrous sulphate tetrahydrate) and Alunogenite (aluminum sulphate) were developed in masses of crystals, almost obscuring the parent rock.

Table I. Sulphate Content of Exposed Roof Strata

Sample	Distance Above Top of Seam, In.	Pct $SO_4^{=}$—S
Shale	5	0.58
Rider coal	21	0.87
Shale	35	1.17
Bony coal	38	2.72
Rider coal	48	0.86
Dense shale	60	0.37

In new workings situated so that the possibility of acid water backing up and coming down as roof drips is remote, acid roof drips were frequently noticed. However, they occurred only where a roof fall had exposed the immediate overlying strata. Water dripping through intact roof and roof coal was not acid. Wet or damp areas containing numerous sulphur balls were not found to be acid. The damp surface of a sulphur ball exposed for 3 weeks to air did not have an acidic reaction. A nearby recent roof break had drops of acid water.

The general results of these observations seem to be that of the many sulphuritic materials associated with the Pittsburgh seam, the most readily and rapidly oxidized are the high sulphur shales and rider coals. The exact nature of this material is not known but it may be finely disseminated pyrite. There is no doubt that sulphur balls, if left underground, in time oxidize to ferrous sulphate. In the Pittsburgh seam, which is the greatest acid producer, the vast extent of potentially acidic roof strata probably is the most important single factor in acid formation. In other seams the roof coals and shales are neither as great in quantity nor as uniformly acid forming. It would be informative to investigate acid forming and nonacid forming shales such as those in the Upper Freeport with a view to chemical composition and the behavior of sulphur oxidizing bacteria.

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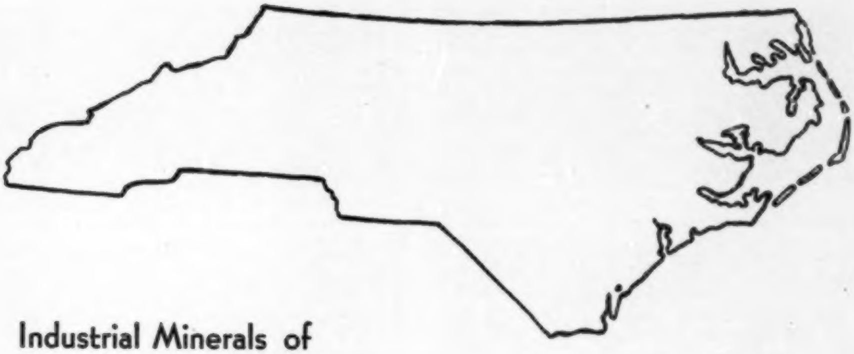

Industrial Minerals of North Carolina

by Jasper L. Stuckey

Geological investigation and research have contributed greatly in making industrial minerals the basis of an important industry in the state. North Carolina contains a wide variety of industrial minerals and rock. Mica, feldspar, kaolin, halloysite, talc, pyrophyllite, sillimanite, and spodumene are used to illustrate the progress being made.

IN a previous paper[1] an attempt was made to indicate the importance of industrial minerals in North Carolina and the wide variety of such materials present. The present paper is concerned with geological investigation and research which have made industrial minerals the basis of an important industry in the state.

Geological mapping has gone through three stages in North Carolina during the present century. The first stage is represented by excellent reports of which *Corundum and the Peridotites of Western North Carolina,*[2] Volume I, now a classic, is representative. In this report, a geologic map of the western third of the state is shown on a scale of 10 miles to the inch. The next stage is represented by folios[3] of the U. S. Geological Survey published between 1902 and 1932. Most of these contained 30 min quadrangle maps on a scale of 2 miles to the inch. The third step is represented by reports prepared during the past 10 years which contain geological maps[4] on scales varying from 500 ft to 1 mile to the inch, while many mine maps have been prepared on scales varying from 20 to 150 ft to the inch.

As the scale on which geological mapping has been done has decreased, the amount of information secured has increased. Pratt and Lewis did excellent work on the peridotites of western North Carolina, but their maps lacked details. Keith and others added

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to the information on peridotites in various folios of the U. S. Geological Survey, but a scale of 2 miles to the inch on a map gives little real information. It was not until detailed mapping was done by Hunter[5] and Murdock[6] that the peridotites were known to contain large deposits of olivine and important amounts of vermiculite, asbestos and talc.

Ries,[7] Keith,[8] Watts,[8] and Bayley,[9] because of a lack of detailed geological maps, stressed the small size of the pegmatites, and Bayley estimated the recoverable kaolin of the state at about 625,000 tons of refined product. The more detailed work by Hunter,[10] Olson,[4] and Parker[11] indicates that at least 225 sq miles in the Spruce Pine district are underlain with a fine-grained pegmatite or alaskite type rock which contains 50,000,000 tons of crude kaolin, large amounts of flake mica and almost unlimited reserves of feldspar.

Research

Investigation and research in mineral dressing and utilization have also played important roles in the development of an industrial mineral industry in the state. The first important work of this type was begun in 1936 when the Tennessee Valley Authority established a ceramics laboratory at Norris, Tenn., to work out improved methods of refining North Carolina kaolin. As a result of experiments carried out jointly with the U. S. Bureau of Mines and the Harris Clay Co. of Spruce Pine, N. C., a modern kaolin refining plant was built in the Spruce Pine district in 1938.

In 1945 a cooperative program was inaugurated under which the North Carolina State College, the North Carolina Dept. of Conservation and Development, and the Tennessee Valley Authority began operation of a Minerals Research Laboratory in Asheville in a building constructed specifically for that purpose by the state of North Carolina. A number of programs of mineral beneficiation have been carried on. Among the more important programs are: 1—recovery of feldspar from alaskite or granite by flotation, 2—bleaching stained talc to improve the color, 3—concentrating flake or scrap mica, 4—drying and grinding mica, 5—concentrating sillimanite, and 6—flotation studies on spodumene.

The industrial rocks and minerals of North Carolina which have become of greatest interest and importance during recent years include: granites and gneisses, clays and shales, sand and gravel, limestones and calcareous rocks, peridotite minerals, pegmatite minerals, and minerals associated with crystalline rocks, such as talc, pyrophyllite, sillimanite and kyanite. Since it is impossible to discuss all of these minerals in detail, the pegmatite minerals, talc, pyrophyllite, sillimanite and spodumene will be used as examples.

Feldspar

Feldspar production in North Carolina began in the Spruce Pine district with one producer in 1911, and by 1917 the state had become the leading producer, which position it still holds.

In the early days of the feldspar industry, production came largely from mica mines and small feldspar mines and prospects. As a result, standard grades and quality control were lacking, and much of the feldspar shipped was unsatisfactory. This led, in the early 1920's, to improved conditions under which larger companies were formed and mills for grinding feldspar were established in or near the Spruce Pine district. With these changes came methods of quality control which led to the abandonment of the smaller dikes and prospects and a search for larger feldspar bodies. It was soon discovered that large dikes with definite schist walls and bodies of feldspar in large masses of pegmatitic or graphic granite contained ample feldspar for enlarging the industry. A number of mines, among them the Deer Park, Hoot Owl, Chestnut Flats, Crabtree Falls and the McKinney, were developed in the Spruce Pine district, from each of which 50,000 to 100,000 tons of feldspar were removed over a period of years. While the major feldspar production came from the Spruce Pine district, worthwhile production has also come from many other mountain counties.

A second major development in feldspar production took place about the beginning of World War II when improved geological mapping demonstrated that large reserves of feldspar are present in fine-grained pegmatite, commonly known as alaskite,[10] in the Spruce Pine district. The knowledge that such materials existed in large amounts and the increased demands for feldspar, especially for use in the glass industry, lead immediately to concentration studies. In 1945 the Carolina Mineral Co., a division of the Consolidated Feldspar Corp., began operating at Kona a modern plant to produce feldspar concentrates by flotation. In 1946 the Minerals Research Laboratory, Asheville, began pilot plant research on the flotation of feldspar. Based on the results of this work, the Feldspar Flotation Corp., Spruce Pine, put into operation in January 1949, a modern $300,000 plant for the production of feldspar from alaskite. The first year's operation was so successful that additional equipment to increase output has already been installed. The United Feldspar and Minerals Corp. also began operation of a feldspar flotation plant in 1949, and plans are already underway to improve and enlarge this plant.

To furnish a feldspar of higher potash content than can be obtained from the alaskite, both the Carolina Mineral Co. and the United Feldspar and Minerals Corp. are adding to their mill feed material from large dumps which have accumulated at some of the older block spar mines in which high grade potash feldspar is abundant. The Feldspar Flotation Corp. is increasing its output of potash material by blending feldspar from alaskite with higher potash content feldspar produced by the Feldspar Milling Co. nearby.

About 40 pct of the alaskite going to the crusher is quartz that is recovered by flotation free of impurities. Much of this was formerly discarded as waste, but most of it is now being sold as potter's flint, and for glass making or for concrete and plaster. Most of the feldspar produced by flotation is for the glass trade with smaller amounts going for pottery. The demands for both uses are increasing. With changes due to production from alaskite, North Carolina has recently become the leading producer of ground feldspar.

Mica

The production of mica began in North Carolina in 1867. At first only sheets large enough to cut into patterns were of interest, but the scrap obtained from trimming the sheets soon became important for producing ground mica. With the advent of kaolin mining it was learned that much flake mica from a few inches to ½ in. in diam could be saved and made good grinding stock. The advent of feldspar mining in 1911 added supplies of both sheet and scrap mica. With increased demands for both kaolin and mica it was determined about 1916 that important amounts of mica less than ½ in. in diam could be saved in the process of washing kaolin. During the early 1920's interest turned to mica schist, and a considerable amount of this material was used for its mica content.

In 1938 it was discovered that many square miles in the Spruce Pine district[10] are underlain with a fine-grained pegmatite or granite type rock, that contains important deposits of partly altered pegmatite rock in which important amounts (10 to 20 pct) of scrap mica are present. Recent geological studies indicate the presence of such materials associated with pegmatites in other districts than Spruce Pine. The mining of these deposits has increased production of ground mica in the state.

Formerly all mica flakes smaller than +8 mesh were discarded in the older type working plants. Investigation in the Minerals Research Laboratory, Asheville, during the past 2 years has shown that approximately 50 pct of all the mica in many deposits is −8 mesh in size. A process, involving the use of Humphreys spirals, has been worked out for saving this fine mica, thereby practically doubling scrap mica reserves of the state. Both wet and dry ground mica are produced, but the improved quality from the washers is making possible a dry ground product which is beginning to compete with the more costly wet-ground mica.

The drying of washed scrap has been a major problem in the industry for years. Driers were first heated with wood, and later by stoker-fired coal burners. More recently oil has become important, and now a strong effort is being made to bring natural gas into the area. Much of the mica is dried and bagged for shipment to the grinding plant. One important producer of washed mica had been shipping by rail. The drying plant of this producer burned, and shipment of wet mica in coal cars was tried. It was found that the vibrations of the cars en route dewatered the mica to such an extent that a smaller dryer at the grinding plant could be substituted for the larger and more costly plant which had burned. While North Carolina has been recognized for many years as an important producer of sheet mica, this industry is unstable and varies widely with economic conditions. Scrap mica, all of which is sold as ground mica, has shown a steady increase in production for many years and has more than trebled since 1940.

Halloysite

In the early days of kaolin mining in North Carolina between 1888 and 1900, some of the deposits in Jackson County furnished a clay that did not conform in all physical properties to true kaolin. Recoveries of as much as 40 pct of kaolin were reported from some deposits. Recent work by Hunter[10] has demonstrated that considerable amounts of that clay was halloysite and also that halloysite is not uncommon in the kaolin deposits of the state.

The presence of halloysite in important amounts in association with kaolin was first established in the early 1940's and halloysite mining actually began in 1943. It is now known that halloysite occurs in two main types of deposits: one of these is block halloysite which occurs as rich pockets in pegmatite dikes, and the other is halloysite mixed with quartz and clay as lenses in the large kaolin deposits in the alaskite or granite bodies. The first type is widely scattered throughout the pegmatite-bearing portions of the western part of the state. The second is restricted to kaolin deposits associated with the alaskite or granite of the Spruce Pine district.

Block halloysite is mined by opencut methods from the pegmatite bodies. The pure halloysite is handcobbed as blocks of varying size which are inspected and hand-trimmed to remove all iron stain. Following this the trimmed blocks are air dried and sold. This process is used only to a limited extent. In the larger kaolin deposits associated with alaskite or granite, the halloysite-bearing lenses are left until enough is secured to justify mining. These lenses are then mined, transported to the clay washing plant, and refined just as kaolin is refined. Halloysite mining and concentrating is costly as compared with kaolin, but the price is higher. Production of halloysite is increasing rapidly in the state.

The fact that detailed geological investigation and research on the preparation and use of pegmatite minerals have brought about rapid advances in their production in North Carolina can be best illustrated by comparing the 1940 production value of these minerals with their January 1950 production value. In 1940 the total value of all pegmatite minerals produced in the state was reported as $1,020,863. Reliable and conservative estimates indicate that the January 1950 production value of pegmatite minerals in the Spruce Pine district alone was at the rate of $6,000,000 annually.

Sillimanite

Sillimanite has been known as a rock-forming mineral in North Carolina since 1873, but it was not until 1943 that deposits of potential value were reported. In that year Charles E. Hunter,[11] TVA Geologist, discovered extensive outcrops of sillimanite-bearing rocks near the town of Valdese and in the South Mountains of Burke County. During the summer of 1945 the occurrences were examined and proved to be more extensive than was expected. A zone approximately 10 miles wide and 95 miles long and extending from near Elkin, Surry County, to Cliffside, Rutherford County, was outlined in which considerable sillimanite was found.

During the following 3 years interest increased in sillimanite, and new deposits were reported. Some concentration studies were carried out in the Minerals Research Laboratory, Asheville, but the results were unsatisfactory partly because of the highly weathered conditions of the samples.

By the summer of 1949 sufficient interest had developed to justify further study. A more detailed examination of all reported occurrence was made,

and samples were collected for concentration studies. In addition to the zone extending from Elkin to Cliffside another zone was discovered which begins in Jackson County, crosses Macon County north and west of the town of Franklin and extends into Clay County.

Two types of sillimanite occurrences were observed. One of these consists of irregular masses of coarsely crystalline sillimanite varying in weight from 1 to 2 lb to as much as 250 or 300 lb. Such masses contain approximately 90 pct sillimanite. The other consists of irregular lenses of sillimanite gneiss and schist that contain from a trace to 20 pct sillimanite.

On the basis of work done to date, both the quantity and quality of sillimanite in the state appear promising.

Talc

Talc mining began in North Carolina in 1859 and has continued intermittently since. A few mines, including the Hewitt, the Hayes, and the Kinsey, were operated for several years with reasonable success, but because of a lack of geological information much useless prospecting was done. In 1945 detailed geological mapping of the area was begun on a scale of 1:24,000.[12] It is now known for the first time that adequate reserves of talc for long term operations are available. A process for reclaiming iron-stained talc has been worked out—further increasing the potential reserves. The talc mining and preparation industry of the state has been recently described by Van Horn.[13]

Pyrophyllite

Pyrophyllite deposits have been known to occur in North Carolina since 1856, but their true value was not realized until the publication in 1928[14] of a report containing detailed geological information on the deposits and a modern geological map of the area. Mines and mills are now being operated at Robbins and Glendon, Moore County and Staley, Randolph County. Pyrophyllite is being mined near Snow Camp, Alamance County and manufactured into refractories at Pamona, near Greensboro. These refractories are being shipped as far as Chicago. Pyrophyllite mining has become of major importance in the mineral industry of North Carolina in recent years and constant search is being made for new deposits.

Kaolin

Modern kaolin mining in North Carolina was begun near Webster, Jackson County, in 1888 and for some 20 years was carried on chiefly in Jackson, Swain, and Macon counties. All the kaolin bodies in that area are in relatively small pegmatites, and mining was carried on from open cuts for the first 15 to 20 ft followed by circular type cribbed shaft for greater depths. Mining was carried out entirely by hand labor. When kaolin mining shifted to the Spruce Pine district, between 1900 and 1910, where there are large deposits derived from alaskite, open-cut mining became standard practice. Owing to the topographic position of many of the deposits, hydraulic mining replaced hand labor in getting the clay into the flume lines. As the pits became deeper, bucket elevators came into general use. In the early 1920's attempts were made to substitute mechanical classifiers for the old machinery, and gradually processes in use today became established.

Today, kaolin mining and concentrating is completely mechanized. At the newest washing plant the clay is ground in Hardinge ball mill and concentrated in a cone classifier. This has increased the recovery by disintegrating the coarser particles, many of which were lost in the older processes, and has made available finished kaolin of a much higher quality than was produced formerly.

Spodumene

It has been known for many years that small amounts of spodumene occur in pegmatite dikes in the Kings Mountain district. Little interest was shown in these occurrences until the publication in 1942 by the U. S. Geological Survey[16] of a report containing detailed geological maps and careful estimates indicating the presence of 650,000 tons of recoverable spodumene in rock averaging 15 pct spodumene. During World War II the Solvay Co. built a flotation plant at Kings Mountain and produced spodumene with some success for the war effort. At the end of the war the plant was closed and has remained idle since. Early in 1949 the Foote Mineral Co. leased mineral lands adjoining the Solvay holdings and later leased the Solvay property. Detailed flotation studies on concentrating spodumene and on iron removal from spodumene concentrates are being carried out in the Minerals Research Laboratory in Asheville. A series of chemical analyses indicate almost a theoretical and an unusually uniform lithia content in the spodumene throughout the district. Plans are being developed to produce spodumene for the ceramic industry as well as for lithium compounds.

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Operational Studies in the Pennsylvania Slate Industry

by W. F. Mullen and C. W. Stickler

WITH few exceptions, unit operations in the Pennsylvania slate industry in 1950 did not differ appreciably from production methods described by Behre[1] and Bowles[2-4] several decades ago. Many traditionally picturesque but relatively inefficient hand operations continued to contribute to high operating costs in an industry in which the margin of profit, for various reasons, was admittedly low.

As part of a general program in which the Pennsylvania State College has been assisting the slate industry to solve some of its problems, time and method studies were used in an effort to determine the bottlenecks of production flow. The results are as applicable to other stone industries as they are to slate.

A study of the basic elements in the production of roofing slate, structural slate, and slate blackboards among selected slate producers revealed comparatively few refinements. Any marked difference in operating method was characterized largely by the available equipment, a noticeable difference in the working properties of the rock quarried by a particular company, and by either the habit thinking or ingenuity of the individual operators.

With the hope that significant economies in manufacture might result, all the unit operations were broken down into elements for study and analysis. The greater part of the study encompassed those operations from the quarry rim to the finished product; however, to obtain a reasonable synthesis of time allotments per operation, several analyses of quarrying from the finish of the wire saw cut are included for reference. The complete study of quarrying made by Bowles several years ago resulted in the introduction of the wire saw into the operation; there appears to be a need for as fundamental a change in the processing of finished slate as the wire saw was in the quarrying process.

Inspect and Measure
Transport Core Drill to Locations
Drill Core Holes for Wire Saw Standards
Mount Wire Saw
Saw Face of Working Block
Inspect and Measure
Drill Plug Holes
Plug and Feather for Floor Break and Required Size
Wedge Slab from Bed
Block up *Style* to Permit Removal
Attach Hoisting Chain
Hoist by Overhead Cableway
Transport to Quarry Rim
Unload Onto Tram Cars or Truck

Fig. 1—Flowsheet of quarrying operation.

Fig. 1 illustrates the flowsheet of quarried rock from the parent bed to the quarry rim with the basic elements which contribute to the difficulty of removing slate from the ground.

In Pennsylvania roofing slate is produced by one of three procedures, many operations of which are quite similar. In the traditional or classical method the quarried rock is reduced by sculping and auxiliary manual treatment to workable size and then is split and dressed (trimmed to size) in the conventional shanty. Archaic as this method might appear, there are several elements that result in substantial savings over more mechanized operations.

In a modified version of this method, the block is reduced by sawing and gouging preparatory to splitting and dressing in indoor stations. This method is particularly adaptable to rock which fractures unevenly by conventional sculping methods.

In the third and more modern method, reduced block received from the block maker is diamond-sawed to length and finished on a production line. Fig. 2 illustrates the flow pattern in each case.

In the modified and more modern plans, dressing is done on mechanically-operated trimming machines, which appreciably reduce the fatigue factor of the operator. However, it should be noted that it is advantageous for the operator to be able to control the speed of the knife to prevent breakage of certain classes of stock. In the classical plan the splitter and dresser normally act as blockmakers also and carry their *eighter* slabs (equivalent in thickness to eight shingles) into the shanties themselves, all of which adds to fatigue and reduced efficiency. The extremely low investment and operating expense of the classical method has undoubtedly been of paramount importance in its continuation.

Production of Mill Stock

The production of mill stock is accomplished by less picturesque methods than enter into the production of roofing slate but it is, in the main, relatively more efficient because of the increased use of machinery for handling and for finishing.

Mill stock can be classified into two categories as far as difference in production method is concerned: structural slate and blackboards. The significant difference in operation is caused largely by the nature of the rock used for each product with the best and easiest cleaved rock being reserved for blackboards to permit the splitting of large slabs to as little as ½ in. in thickness. Structural stock, on the other hand, is split into panels no thinner than 1½ in. and

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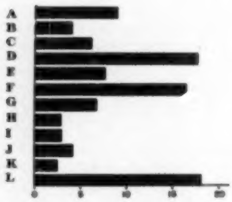
Table I. Quarrying

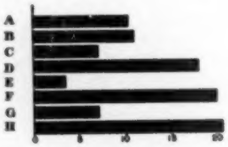
	Element	Average Time, Min
A	Inspect and measure	2.35
B	Drill plug holes	1.05
C	Plug and feather	1.57
D	Wedge slab from bed	4.65
E	Style	2.00
F	Signal and wait for tackle	4.35
G	Attach chain to tackle	1.73
H	Position block	0.72
I	Hoist, 75 ft	0.75
J	Transport to rim, 100 ft	1.02
K	Unload	0.61
L	Incidentals, look for and put away tools, wait for help, remove waste, etc.	4.75
		25.55



Relation of elements to unit operation, pct.

Table II. Block-Making

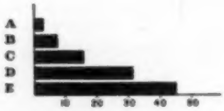
	Element	Average Time, Min
A	Inspect and measure	3.71
B	Drill plug holes, three	4.03
C	Line holes	2.49
D	Plug and feather	6.58
E	Bar and turn over	1.24
F	Split into eighters	7.33
G	Carry to splitting station	2.58
H	Incidentals, swab slab, look for tools, trim clean-up, etc.	7.06
		35.02



Relation of elements to unit operation, pct.

Table III. Splitting 3/16 In. Shingle

	Element	Average Time, Min
A	Pick up eighter	0.05
B	Split to ½'s	0.16
C	Split ½'s to ¼'s	0.33
D	Split ¼'s to ⅛'s	0.68
E	Measure shingle, pick to desired length, and remove to storage for dresser	0.97
		2.19 per eighter

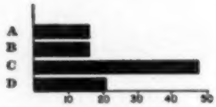


Relation of elements to unit operation, pct.

Table IV. Dressing

	Element	Average Time, Min
A	Pick up slate	0.03
B	Make* slate	0.03
C	Measure and trim	0.09
D	Stack on rack	0.04
		0.19 per shingle

* Determining the proper size finished shingle that can be obtained and making the initial trim on the dressing machine.



Relation of elements to unit operation, pct.

normally is characterized by a rough split, which requires a planing operation in addition to the surface treatment given to blackboard stock. Considerable attention was devoted to the planing operation during our study inasmuch as it appeared that fundamental improvements in the milling sequence would hinge largely on the planing operation as a nucleus.

Fig. 3 illustrates the flow pattern for both structural slate and blackboard production. Under the category of *special finishing* of structural slate are included such manual operations as chamfering of edges, sawing or planing of grooves, and the drilling of holes.

Supplementary storage and consequent rehandling operations not shown on the flow pattern occur frequently, particularly when normal flow is interrupted and slabs accumulate at a given station. The handling problem is particularly acute inasmuch as the only methods currently employed are those involving trundling, carrying on the operator's back, wheeling on a two-wheel cart, or, in a few exceptional cases, the use of a manually operated pneumatic hoist.

Time Study

The time study was employed as a means of analyzing the present production pattern for purposes of determining the contributory factors in the present relatively inefficient methods for finishing slate. There was no intent to establish standards or to become involved in the various ramifications of cost control. The purpose of the data is to direct attention to those elements or operations that appear to be most in need of correction; future and more detailed studies can be employed to complete the picture.

Tables I through VIII show representative time for the individual elements of each unit operation in addition to showing the percentile relationship of each element to the total operation and each unit operation to the total amount of work required to produce the finished product.

More detailed analysis of each element and operation falls outside the scope of this discussion but is contained in a forthcoming bulletin of The Pennsylvania State College Mineral Industries Experiment Station.

Tables IX and X tabulate the comparative amount of time required per unit operation. Table IX covers the production of 160 roofing shingles from an orig-

Table V. Sawing

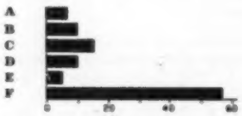
	Element	Average Time, Min
A	Load	6.10
B	Unload	1.15
C	Brace	4.32
D	Wedge sawed sections	7.85
E	Saw four sides	75.40
		94.82

Relation of elements to unit operation, pct.

Table VI. Planing

	Element	Average Time, Min
A	Load	0.94
B	Brace	1.50
C	Incidentals[a]	2.30
D	Turn over	1.50
E	Unload	0.60
F	Plane both faces	8.86
		15.60 per slab

[a] Align knife, adjust machine, clean up, etc.



Relation of elements to unit operation, pct.

inal block 172 in. long, 32 in. wide, and 9 in. thick with transport time from quarry rim to station and punching time for roofing shingles included to complete the study. Table X covers the production of six structural slabs each 33½x53x¾ in. thick from a block sawed on four sides to these linear dimensions. The original thickness of this block before splitting was 9⅞ in. Transport time from quarry rim to mill and splitting time for the slabs are included to complete the study.

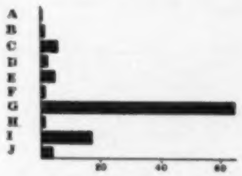
Analysis of Results

A study of the various time summaries obtained reveals one troublesome factor: each block removed from the quarry possesses inherent characteristics that affect the handling methods and operational time. In planning flow patterns it is essential that methods be devised to nullify these variations through improved mechanical equipment or to allow flexible routines in the milling operations.

Handling time poses problems in the way of distance traveled, weight and size of block, condition of roadbed, and type of hauling unit. Weather conditions and safety regulations also play dominant roles in anomalies from normal conditions. Such variations appear to be sufficiently frequent to preclude any standardization from data obtained at this stage of the study. As yet no reproducible relationship has been formulated to apply to the many plants studied in the industry but the results of our study have indicated definitely the possibilities of

Table VII. Sand Rubbing

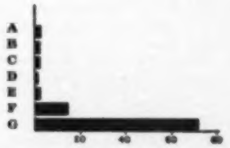
	Element	Average Time, Min
A	Position slab horizontally	0.25
B	Place weights on slab	0.49
C	Change weights	1.10
D	Reposition board	0.42
E	Gage and remove weights	0.87
F	Turn over slab and replace weights	0.34
G	Rub faces	15.86
H	Position slab vertically	0.33
I	Rub edges	4.11
J	Unload slab from bed	0.77
		24.53 per slab



Relation of elements to unit operation, pct.

Table VIII. Polishing

	Element	Average Time, Min
A	Transport slab to polisher	0.34
B	Position slab on table	0.30
C	Touchup	0.25
D	Unload from machine	0.19
E	Remove from storage	0.29
F	Wash and inspect	1.37
G	Polish	7.49
		10.23 per slab



Relation of elements to unit operation, pct.

Table IX. Production of Roofing Slate

Unit Operation	Time, Min
Quarrying	25.55 (1 block)
Transport	9.70 (1 block)
Block-making	35.62 (20 eighters)
Splitting	43.80 (160 shingles)
Dressing	36.40 (160 shingles)
Punching	15.98 (160 shingles)
Total	167.06

improved methods, equipment, and routines that may be applicable to the industry as a whole.

Suggested Improvements

Based on the observations gathered in this study, certain conclusions were reached with respect to methods improvement, equipment design and better utilization of raw material.

It is apparent that major changes are necessary in present operating methods not the least of which would be the finishing of multiple pieces in one operation. These unit operations under consideration for this transformation are: 1—splitting, 2—dressing, 3—sawing, and 4—planing, rubbing, polishing. Composite machines to do the work of several opera-

Fig. 2—Flowsheet of roofing slate production.

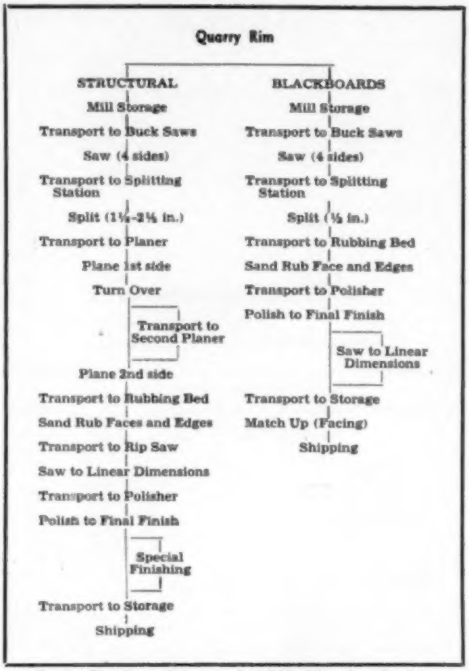


Fig. 3—Flowsheet of structural and blackboard production.

tions would materially decrease the amount of handling time between operations and reduce the amount of waste resulting from separate operations. Under study at the college are new designs in planing, rubbing, and polishing operations that will simplify material flow, increase the yield from quarried slate, and increase production.

There are promising indications that saw speed with the present design saw can be increased at least fourfold by the use of a new type inserted tooth now under test. Sawing with the present saw occupies such a prominent relationship to the overall time picture mainly because only one side of a block can be sawed at one time. Bowles[3] has referred to the use of gang saws in the Vermont-New York region for slate too hard to cut by conventional saws. Gang-sawing may be the answer to multiple side sawing of blocks although it may not necessarily take the form of the conventional bladed gang saw.

Table X. Production of Mill Stock

Unit Operation	Time, Min
Quarrying	25.55 (1 block)
Transport	6.13 (1 block)
Saw (4 sides)	94.92 (1 block)
Split	19.06 (6 slabs)
Plane	93.60 (6 slabs)
Rub	146.58 (6 slabs)
Polish	61.38 (6 slabs)
Total	447.22

Under consideration at the present time are two possibilities: a multiunit wire saw for sawing two sides of the block simultaneously either with single blocks or blocks arranged in tandem (the French[5] have used wire saws for their milling operations rather successfully) and a multiunit diamond saw with speed reducer to control the load on the motor with the nature of the rock and the speed of sawing. Another possibility that could be developed to minimize the need for mill sawing would be the design and development of a quarry saw on the chain-saw or cutter bar pattern, which could saw blocks to mill size directly from the bed. In addition to eliminating the need for mill sawing, such a saw would eliminate most of the waste resulting from the ragged fractures, which are sawed off and discarded in the mill.

Also under study is the use of overhead monorail conveyor systems in the mill as a corrective for handling delays.

Further industrial studies appear to be necessary to expand present information on slate production. Sufficient attention to this overlooked industry should soon result in a production pattern that will permit lower operating costs and greater prosperity for the industry.

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Mine-Drainage Studies in the Iron Ranges of Northern Michigan

by Wilbur T. Stuart

THE increased demand for iron ore has necessitated a re-examination of ore-bearing lands on which the presence of water previously has indicated hazardous and expensive operating conditions. In view of the importance of iron ore production to the national economy and defense, the Ground Water Branch of the U. S. Geological Survey, in cooperation with the state of Michigan, began a study of mine drainage in the Iron River district in 1945, and later extended the work to the Marquette district. The purpose of these studies was to define the principles involved in the movement of surface and ground water toward the mined areas, with the hope that the information obtained in the research would lead to the development of methods of water control and to a reduction in the total mining costs.

Not only are the direct costs of drainage increasing, but also the indirect operational costs of working a wet mine are becoming a larger proportion of the total mining costs. For some wet mines the direct costs of pumping and drainage may range from 35 to 40¢ per ton of ore produced, but the indirect costs due to handling wet ore and controlling the water may be five to ten times this amount.

The methods of study were formulated as the work progressed, and inasmuch as they were the first large-scale studies of their kind, they should serve as a guide for the solution of similar problems in other mining areas. The Iron River district was chosen for a pilot study because in this district the longest records of mine pumpage and water-level observations were available, including, as they do, the records for the Homer mine of the M. A. Hanna Co., where pumping from surface wells began in 1930.

The results of the first investigation by the Michigan Department of Conservation have already appeared.* The first section of the report on a similar study of the Marquette district, which was started in 1948, will be published this year.

Methods of Study

In an effort to reduce the flow of water into the mine workings in the Iron River district, about 4500 gpm was pumped from the bedrock being mined and about 9000 gpm from the glacial overburden. In the Marquette district in the vicinity of Ishpeming and Negaunee, about 5000 gpm was pumped from the bedrock and about 4000 gpm was pumped from the glacial overburden. Where the water was pumped only from the bedrock, the rate of pumping ranged from a few hundred gallons per minute in the dry mines to many hundred gallons per minute in the wet mines. In each district, pumping from the overburden was localized on a few properties where costly pumping installations had been made and the expenditures for power had been large.

In each district a comprehensive ground-water investigation was made of the whole area, involving the collection and interpretation of all the available data bearing on the source and quantity of water to be controlled. Although it is not the purpose of this paper to discuss the methods of making a ground-water investigation, it should be pointed out that a drainage study follows a pattern of engineering analysis that determines the occurrence, source, movement, disposal, and quantities of water involved. The investigation in the iron-mining districts of Michigan began with the construction of a map of the buried bedrock topography. Because the ground-water reservoirs occupy the low points in the bedrock basins, this map gives information concerning their areal extent, depth, shape, and degree of interconnection. The depth to water in the drillholes and wells indicates the altitude to which the ground-water reservoirs are filled, and the logs of the material penetrated in the drillholes and wells indicate the general character of the materials filling the reservoir. The slope of the ground-water surface indicates the direction of flow through the reservoir, the water movement being from the points of higher altitude to points of lower altitude. By analysis of the rise and fall of the ground-water levels in response to additions of water through recharge and to changes in the rate of discharge through pumped wells, estimates of the total quantity of water in storage and of the rate of flow through the reservoir

* W. T. Stuart, C. V. Theis, G. M. Stanley: Ground-water Problems of the Iron River District, Mich. Technical Report No. 2, Geol. Survey Div., Mich. Dept. of Conservation.

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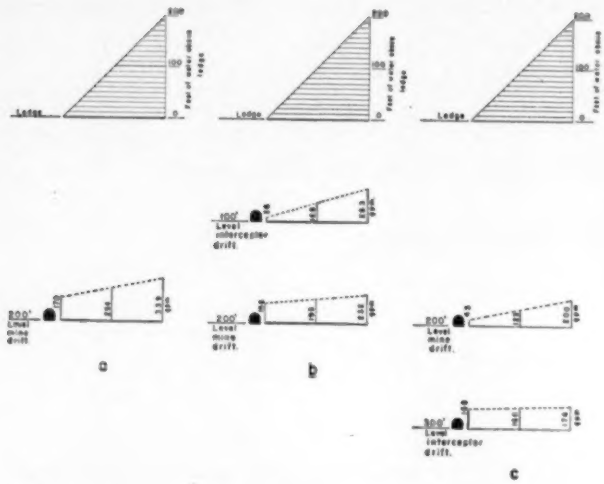


Fig. 1—Relative discharge, in gallons per minute, from mine and interceptor drifts 1000 ft long, 16 ft wide, at footwall, at selected levels in a vertical (iron formation) aquifer 157 ft wide.

can be made. When the principles of movement of the ground water to the mined areas are understood the most feasible means of control can be selected.

Recharge

The ground-water reservoirs in northern Michigan are replenished or recharged by precipitation and in certain areas by the induced percolation from the surface streams and ditches. Through studies of the records of the water-level fluctuations in the wells and drillholes it has been determined that approximately 20 pct of the total precipitation recharges the ground-water reservoir, which represents, in a normal year, an equivalent of about 200 gpm per sq mile. In the Mineral Hills area of the Iron River district the recharge induced from the surface streams and ditches within the cones of pumping depression may exceed an average of 1000 gpm. In the Marquette district the recharge induced from the Carp River near the Morris mine of the Inland Steel Co. probably averages about 500 gpm.

Ground-Water Movement

In the zone of circulation above the bedrock the ground-water movement is from the points of high altitude, where replenishment is taking place, to the points of low altitude where the ground water is discharged at the land surface. The gradients are normally low, and the velocity of water is also low, even though the water-bearing materials may be able to transmit a large amount of water. When the altitude of the discharge point is artificially lowered, such as in the cone of depression around a pumping well or other drainage structure, the gradient is increased greatly, and the quantity of water flowing to the structure is increased greatly. When an active mine lowers the water level through mining operations, the gradient is increased, and a cone of water-table depression or a ground-water sink is formed in the saturated glacial overburden above the point at which the water is descending into the mine workings. The drainage downward through this ground-water sink produces the same hydraulic result in the glacial overburden as a vertical well discharging an equal quantity of water, and the same mathematical formulas apply in both cases to the rate and extent of the cone of depression. These formulas indicate that the depth of the cone varies directly as the quantity of ground water withdrawn; and the increase in diameter of the cone varies directly as the time since the withdrawal of water began, although the outward growth is faster when the process takes place in minerals of greater permeability.

The investigations in the Michigan mining districts show that pumping the ground-water reservoirs in the glacial overburden has little effect on the flow of water into the bedrock, so long as there is a great thickness of water over the bedrock openings. Pumping from the overburden is most effective where the areas of permeable bedrock can be dried up by drawing the water table down so as to cause the ground-water shore line in the overburden to retreat down the bedrock slope, either uncovering the bedrock or reducing the head of water above it to a negligible amount. It was further determined during the investigation that, where dewatering is attempted by means of wells, they should be located as near as possible to the point at which the water enters the bedrock.

There are two manners in which water from the ground-water reservoirs in the glacial overburden can move into the bedrock: 1—by slow percolation through the capillary openings in the bedrock, 2—and by rapid movement through supercapillary openings. In the first case the rate of flow of water through the bedrock itself governs the quantity of water in the mine, and generally the flow is of the order of a few hundred gallons per minute. In the second case, where there is rapid movement in supercapillary openings such as vug systems or cracks formed by the collapse of the roofs of mine workings, the rate of flow into the mine may be governed by either the rate of flow in the bedrock or, in case the openings are of larger size and of greater number, the rate at which the overburden can transmit water to the area. In either case, because of the rapid movement through large openings, the mine would be wet and the quantity of water pumped from it would be many hundred gallons per minute.

In the Michigan iron mining districts the term iron formation is applied to that particular structural member or bed which carries the minable per-

centages of iron ore. In the Iron River district the unoxidized part of the formation consists of interbanded chert and siderite, and the oxidized part consists of chert interbanded with soft red and yellow oxides, mostly hematite. In the Marquette district the unoxidized part of the formation consists of iron-bearing slates, schists and cherts, and siderites, and the oxidized part consists mostly of hematite with small widely disseminated particles of magnetite. In both districts the footwall and hanging-wall formations are more resistant members and are not ordinarily considered to be water yielding. Both the footwall and hanging-wall formations in the Iron River district are slate members, whereas in the Marquette district only the footwall is a slate. In the Marquette district the hanging-wall may be either a quartzite or a conglomerate.

A study of the movement of ground water in the orebodies indicates that the water enters the iron formation through its surface of contact with the overlying glacial blanket. This is shown by several lines of evidence. First, most of the water enters the mines through the workings in the iron formation. Water entering the drifts of the footwall and hanging-wall strata is of minor amount. Second, the water in the operating mines is of the same chemical character as the waters in the overburden, whereas the waters in the hanging-wall and the footwall strata are more mineralized and of a different chemical character. Third, those mines where the iron formation has its surface contact with the overlying glacial blanket either above the general ground-water level or in areas of small ground-water drainage are relatively dry mines.

The theoretical problems involved in the movement of ground water in the bedrock are more complex and of a different nature from those in the overburden. They differ in principle in that the movement in the bedrock here considered is in a bed or an aquifer that stands vertically, or steeply inclined, whereas the movement in the glacial overburden is essentially horizontal. The water entering the iron-bearing formation through its surface contact with the overlying glacial blanket moves downward essentially under gravitational force. The movement is complex, because the structural control is complex. In many places the effective boundaries of the ground-water reservoir above the iron formation and the structure of the iron formation from the bedrock surface down to the mine levels are unknown. Even where these factors are known they are often too complex to be amenable to mathematical treatment without considerable simplification. Such simplification of complex data requires analysis to obtain feasible approximations of actual conditions. However, one does not throw away a useful tool simply because one wishes it were better.

Some indication of the magnitude and complexity of the problems involved may be afforded by considering the movement of water in a new mine drift or an opening in an iron formation in a new locality. Presumably before the opening is made the water moves slowly. It enters the iron formation through the bedrock surface and moves to a lower point under a slight hydraulic gradient equal to the difference in the altitude between the two points divided by the total length of the formation between them. When the opening is made, a rapid movement of water begins. If the glacial drift blanket is saturated so that the iron formation is furnished all the water it can transmit and the dip of the formation is constant between the mine workings and the bedrock surface, the water will move under a hydraulic gradient equal to the sine of the angle of dip. Mathematical equations have been set up to determine the pressure head and direction of flow of water in an orebody under some of these conditions. These equations indicate that the water descending through the formation is under considerable pressure near the level of the lowest mine development and that the pressure head diminishes greatly at some distance above. The equations furnish a means of estimating the water to be expected in workings at several levels, especially in water-drainage drifts intended primarily to intercept descending water in the formation at a higher level than the main mining operation. In general, for nearly vertical formations it appears desirable to construct exploratory or preliminary openings and to collect most of the water at lower levels, for the cost of lifting water from a lower level is offset by the economy of mining a drier orebody above.

In case of an inclined or dipping formation, other equations are needed, as the pressure head is either increased or decreased by the gravity component of the downward movement of water which directs the flow toward the drift opening. In a monoclinal or rolling formation it is possible for a properly located drift at an upper level to divert all the water from the formation, especially if it is located along the saturated footwall of the structure.

Water Control Methods

Most simple cases of ground-water flow can be approximated mathematically, but others require long tedious computations. Since most mine operators are interested principally in how to control the water and in the amounts that must be handled, the quantities of water that could be expected from the two common methods of drainage or control are given in Figs. 1 and 2. The equations for the calculations are discussed in Mich. Geol. Survey Technical Report No. 2; pages 37 to 41 for Fig. 1, and pages 46 to 50 for Fig. 2.

Fig. 1 shows the flow of water that might be expected from mine drifts in the bedrock with various thicknesses of water in the overburden. In case *a* with a mine drift 200 ft below the bedrock, the flow of water doubles when the head of water in the overburden is increased from a negligible thickness to 200 ft. In case *b* with a main operational drift 200 ft below the bedrock surface and a water interceptor drift 100 ft below the bedrock surface (water is withdrawn from both drifts) the interceptor has little effect on the lower drift when there is a negligible thickness of water in the overburden; but with 200 ft of water in the overburden the total water at the operations level is decreased about 30 pct. The general efficacy of intercepting galleries varies greatly with the height of water above the bedrock. In case *c* in which the interceptor is placed at the 300-ft level and a negligible thickness of water above the bedrock is assumed, the water in the main operations level at 200 ft is reduced about 75 pct although the total cost of lifting the water is increased. Thus it is apparent that exploratory openings at lower levels would produce drier working conditions for mining operations above.

Fig. 2 shows a cross-section of an idealized buried valley being dewatered, and an iron-formation sub-outcrop below the overburden. This cross-section, with its included tables, graphically demonstrates

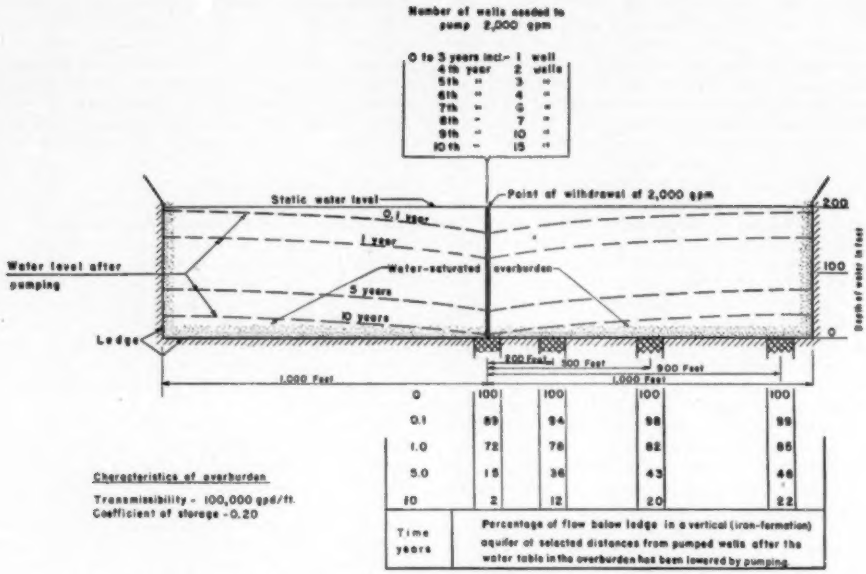


Fig. 2—Cross-section through an idealized buried valley showing relation of water table in the overburden to flow in formations below ledge.

typical events that occur whenever the overburden is dewatered by means of wells.

To indicate the percentage of reduction in flow through a vertical iron formation below a bedrock valley, at the well, and at distances of 200, 500, and 900 ft therefrom, a table of percentage of the original flow is shown for each time period. The greatest reduction in flow takes place where the water is lowest, and this point is over the point of entry of the water—the suboutcrop of the iron formation.

In the computations for Fig. 2 there was assumed to be 200 ft of saturated overburden above the bedrock and a constant withdrawal of 2000 gpm from the valley. No trouble would be experienced in maintaining this constant rate of withdrawal during the first 3 years of pumping, but after that time the decreasing thickness of saturated materials would require installation of additional wells to maintain the 2000 gpm rate. In most pumping installations the well systems dewater the area until the reduced yield results in no further lowering of the water table. Additional wells added to the system result in additional lowering and additional reduction in yield per well unit, and again the system reaches a new equilibrium. At the top of Fig. 2 a table indicates the approximate number of wells needed to maintain a constant yield of 2000 gpm which will lower the water table according to the time periods shown. For simplicity in this figure, it was assumed that all of the newly added wells would be located at the point of withdrawal; although the actual construction of the new wells either closely spaced in a line or a small circle would slightly modify the profiles.

Pumping directly from the glacial overburden requires the installation of expensive surface pumping equipment in an area beneath which mining operations are taking place, and where there is constant danger of damage to or loss of the installation by subsidence. A possible solution would be to encircle the critical area, in which the ground water enters the bedrock, with a ring of closely spaced small-diameter, low-cost wells. Pumping from these wells would lower the ground-water level in the reservoir almost to bedrock within the circle and thus reduce the water available for downward percolation. Because the wells must be closely spaced to reduce the saturated thickness of the water-bearing materials between them, the yield of any individual well would be small, thereby eliminating the need for expensive large-diameter wells. This low cost feature is important, because of the large number of wells needed to encircle the area and the likelihood that some may be lost owing to subsidence.

Other means of controlling water flows such as grouting, chemical seals, asphalt barriers, and freezing have not been tried on a large scale in Michigan, to the knowledge of the writer. However, in shaft sinking and in foundation work all have been used more or less successfully. Probably the high cost of the product, the large amount of drilling needed, and the unwillingness of any contractor to guarantee results have been the biggest factors in restricting the use of these methods, although it is believed that in the future these methods of control will be used with success.

After 5 years of study it is apparent that no one method or even combination of methods can be applied economically to eliminate all the ground water in mine workings. Each problem requires separate study and solution. Some properties may be effectively dewatered by pumping from the overburden. In others where surface pumping is not economical, properly located water-interception drifts may control the water so that a drier ore may be mined with fewer hazards and at less cost.

US Tin Mission To Study Costs in Far East

A move to obtain adequate supplies of tin at prices the United States is willing to pay was initiated when the interagency tin mission left for the Far East.

The Mission, which includes representatives of the Reconstruction Finance Corporation, State Department, Defense Minerals Procurement Agency and the Bureau of Mines will go to Indonesia and Malaya.

The primary purpose of the mission, according to officials, will be to study tin production costs, not to negotiate for the purchase of tin. The mission will be prepared to enter into negotiations, if the Far Eastern producers are amenable.

The United States withdrew as a large-scale buyer of tin some months ago in protest against what it considered exorbitant prices being charged in the Singapore market. Subsequent attempts to negotiate for a steady tin supply at what this country considers reasonable prices have been unavailing.

The U. S. is prepared to enter into a two or three year contract with the price of tin somewhere between $1.00 to $1.12 per lb.

The decision to abstain from the market has not yet put a squeeze on U. S. supplies. Substantial quantities of tin still are coming in from Malaya under an old contract.

Metals Mines Face Manpower Shortages

The metal mining industry faces a shortage of workers, according to the Department of Labor's Bureau of Statistics. Even though the manpower supply is expected to be generally adequate in the nation as a whole, shortages are likely to continue in this industry because of the limited supply in mining communities and because of the sex and age restrictions on mining employees. By 1955 metal mines will need about 120,000 workers, 15 pct more than are currently employed.

The report states that of the 105,000 workers engaged in metal mining in June, 1951, iron mining employed 37,000, copper mining 28,000, and lead-zinc mining 21,000. The remaining 17 pct of the workers are engaged in silver, gold, bauxite, and other metals.

Mechanization has enabled the mining industry to produce much larger quantities of metals than it did 40 years ago with about half as many men. Because of progressive depletion of ore deposits, however, output of metal per man-hour in recent years generally has increased slowly and actually has declined in lead-zinc mining. The nature of new ore discoveries are such that the downward trend in the metals produced per man-hour in the lead-zinc mining will probably hold true for the entire industry.

Mining News Fronts

• The Export-Import Bank announced approval of a loan of $1 million to Mauricio Hochschild of the Sociedad Anonima Minera Industrial to assist in financing the expansion of the Bolsa Negra tungsten mine in Bolivia.

The mine has been a producer of tungsten for a number of years and has extensive ore reserves. The loan will be used to purchase mining and milling equipment in the United States. This equipment is to increase the production of the mine. The mining company will also make a substantial investment in order to complete the expansion program.

• The Anaconda Copper Mining Co. will build a $46 million aluminum plant at Kalispell, Mont. It is understood that Anaconda, subject to government approval, will acquire commitments from the Harvey Machine Co. to build a primary aluminum plant with an annual capacity of 54,000 tons. The new plant will be located about 250 miles north of Butte, the location of some of Anaconda's major operations.

• The Tennessee Valley Authority has signed a new coal contract under two remarkable circumstances. It is the largest single purchase of coal in the history of the world. The order calls for the delivery of 18 million tons of coal in the next ten years, at a cost of $63 million. In the second place, this history-making purchase was made to supply fuel for generating electricity in what was intended originally to be a great waterpower project.

• Ford Motor Co. and the Cleveland-Cliffs Iron Co. soon will, in a joint venture, begin mining operations to produce 400,000 tons of concentrated iron ore a year from the Marquette Range at Humboldt, Michigan, on a property which has lain dormant to commercial mining since 1920. The crude ore will be mined from an open pit and passed through concentrating units which will produce a product substantially higher in iron content than the ores now being commercially mined in the Lake Superior region. First of the concentrating units will be capable of producing 200,000 tons annually. Development of the property will begin immediately and capacity production is expected by late in 1953. A second unit of similar capacity is scheduled to be in operation by 1955. A new company, owned jointly by Ford and Cleveland-Cliffs, will be formed to undertake the development and operation of the property which will be known as the Humboldt Mine. Approximately 1,100,000 tons of direct shipping ore were mined from the Humboldt property between 1865 and 1920.

• Vanadium Corp. of America has leased from Hetzer Mines, Inc. the latter's tungsten mill located at Nederland, Colo., it was announced recently by William C. Keeley, Vanadium's president. Vanadium Corp. will immediately enlarge the capacity of the mill in order to process tungsten ores in that district. A purchase schedule has been set up whereby Vanadium Corp. will purchase ore from local sources.

• Development of magnesite or magnesium carbonate deposits in Brazil by an American company will soon get underway. Earl A. Gardner, president, Harbison-Walker Co., announced that his firm was organizing a subsidiary to work the deposits. The Harbison-Walker firm produces materials used in making industrial furnaces.

• Sil-Van Consolidated Mining & Milling Co. is placing orders to outfit a new 150-ton mill at Hudson Bay Mountain, 10 miles from Smithers, B. C. The concentrator will mill lead-zinc ores carrying gold and silver.

J. J. Forbes Heads U. S. Bureau of Mines

John J. Forbes, one-time miner and a veteran of nearly 37 years with the Bureau of Mines, will succeed James Boyd, who resigned as bureau chief recently. Prior to his present appointment he was chief of the Health and Safety Division. His first job in the Bureau was first aid miner and he slowly climbed the ladder to top man in the Bureau. Mr. Forbes is the first career employee of the Bureau to be named director. John L. Lewis, United Mine Workers, indorsed his selection as "an excellent one."

Upgrading Coal May Furnish Coking Quality

Tests by the U. S. Bureau of Mines indicate that some coal beds in the Fayette County area can be upgraded to coking-coal standards by mechanical cleaning.

With coal reserves in the famed Pittsburgh bed of the Fayette County field nearing depletion, the discovery is an important one. The Pittsburgh bed represents more than 93 pct of Fayette County's coal output. Estimates place its recoverable reserves at approximately 110 million tons.

In searching for substitute metallurgical coals, the bureau tested 17 mine samples from seven beds. According to a Bureau of Mines spokesman, parts of all these beds can be upgraded.

Although tests made to date have not as yet indicated how great a reserve of metallurgical coal mechanical cleaning can produce, one department believes preliminary results are promising.

Glidden, U. S. To Share Drilling Costs

The Government has entered into an agreement with Glidden Co. to share equally the cost of exploratory drilling on the company's zinc properties in Shasta County, Calif.

Under a contract signed by Dwight P. Joyce, Glidden president, and the Defense Minerals Administration, Glidden agreed to start diamond drilling operations on the property within sixty days. Cost of the development work was not given.

Zinc, a strategic metal, has been in critically short supply for the past year. The Defense Minerals Administration is eager to increase domestic output for defense program requirements.

The Government will allow Glidden to use Government barges on the Shasta Lakes to move in drilling equipment.

Previous estimates indicated the properties contained about 225,000 tons of zinc ore. If the exploratory drilling confirms this estimate, underground mining operations will begin at the earliest practicable date, Mr. Joyce said.

AIME Joins Pan-American Mining Men at Mexico City Technical Meeting

The First Inter-American Convention on mineral resources, sponsored jointly as the 172nd general meeting of AIME and the Third Congress of the Pan American Institute of Mining Engineers and Geologists, was held at Mexico City from Oct. 29 to Nov. 2, 1951. The theme of union between the Americas was cemented by bi-lingual technical sessions, exhibitions of the culture of Mexico, and entertainment featuring the best in music and dancing from the Americas.

Some 439 registered for AIME meetings, with the AIME ladies registration totalling 311. The total registration of AIME and IRIMIGEO was about 2000. There were 20 technical sessions, of which nine were held by the Minerals Beneficiation Div. and the Industrial Minerals Div. of AIME.

The del Prado hotel was the headquarters of the meeting. The opening session, addressed by Miguel Alemán, President of Mexico, was held at the Palace of Fine Arts on October 29. This was followed by a banquet at the Military Casino. Afterwards there was a guided tour of Mexico City, climaxed by a colorful pageant of the costumes and dances of the States of Mexico at the del Prado.

Technical sessions held on October 30 were preceded by a Rancheros breakfast sponsored by MBD. Tequila replaced Scotch as the main condiment in this now famous ritual. In the evening, a reception was given at the University Club for the entire convention by the Petroleos Mexicanos. This was more than a reception as a buffet garnished with the delicacies of Mexican cuisine was served. The ladies were feted at a luncheon and fashion show on October 29.

Photographed watching the festivities at the PEMEX reception were, left to right: W. M. Peirce, AIME President; G. P. Serrano; A. J. Bermudez; W. E. Wrather, AIME Past President.

Technical sessions were held on Wednesday and the MBD sponsored a luncheon at the American Club. The final banquet was held at the El Patio night club.

On Thursday, November 1, delegates went to a special fiesta at Xochimilico, the floating gardens.

Two excellent field trips were held on Friday; one to Taxco, the famous old silver mining center, as guests of the American Smelting & Refining Co., and the other to Pachuca as guests of the Compañia del Real del Monterey Pachuca.

Credit for this fine convention goes to Ing. Raul de la Pena, director general of INIRM; A. Terrazas, chairman of the Mexico City Section, AIME; William Kane, general chairman AIME meeting; Harry Beggs, G. A. Golson, and Jack Carty for registration and arrangements; and Carl Fries, program chairman. Senora Terrazas, chairman, Mexico City WAAIME directed the ladies activities. *Photographs of the meeting will be published in the January issue.*

Headquarters Business Methods Studied and Changes Recommended

Recommendations for ways to improve the functioning of the New York headquarters office of AIME were made in a report presented to the Board of Directors at the Los Angeles Meeting on Oct. 25. At the May 9, 1951 meeting of the Executive and Finance Committees, Gail Moulton had been asked to collaborate with Secretary Robie in obtaining an independent expert to review procedures at AIME headquarters. At the June 13 meeting of the Board, it was reported that Herbert H. Vasoll, of the firm of Davies & Davies, certified public accountants, would make the study.

Highlights of the recommendations included (1) Immediate purchase of a modern Addressograph machine to replace present equipment, at an estimated cost of $2500. Such purchase was authorized by the Board. (2) Installation of a "Keysort" punch card system for membership records, at an initial cost of some $600. (3) Purchase of a Pitney Bowes postage meter at a cost of about $600. This was also approved by the Board for early action. (4) Engagement, from without the present organization, of a Controller and Business Manager, who would have supervision over all functions of the Business Office. (5) Reorganization of the office into four primary departments, each supervised by one individual who in turn shall be responsible to the Secretary of the Institute.

These shall be: (a) Institute Activities Dept. Convention planning, meeting arrangements, travel accommodations, membership control and promotion, special projects for the Secretary, public relations, and personnel. (b) Metals Branch Dept. (All activities pertaining to the Metals Branch of the Institute). (c) Publications Dept. (Editorial, Transactions, advertising, production, circulation promotion, publication of proceedings volumes, and any other Institute publications). (d) Business Office. (Accounts, Addressograph, orders, purchasing, filing, and mailing.)

The present Office Manager would be named Administrative Secretary of the Activities Dept. The Metals Branch Dept. would continue more or less as at present except that its functions dealing with the publication of volumes would be placed under the supervision of the Publications Dept. The Publications Dept. would be relieved of all student relations work

and through installation of a Keysort system would be enabled to obtain more complete data regarding readers of the journals, and more readily than at present. Revision of the work of the Business Office, with a new head would permit several improvements in operation. In the Accounting Div., improvements are suggested in billing procedure and in cooperation with other departments which should eventually result in a reduction of staff. The Mail Room procedure would be revised to improve efficiency and to avoid delays, in several suggested ways. The Addressograph Div., would be divested of miscellaneous clerical duties and would become strictly a mechanical service unit, with new equipment, avoiding present mechanical breakdowns. All orders would be processed through one order clerk and all billing done by the Accounting Div.

Some revision of forms was suggested, and installation of a Multi-Lith machine in place of the present Mimeograph equipment. For publication of the Directory, installation of a Remington Rand Flexoprint system was felt worthy of investigation.

As to personnel, "in general we believe that the size of the headquarters staff is commensurate with the volume of work handled. However, due to unequal distribution of work, the services of certain individuals are not being utilized to the fullest extent and others with a greater sense of responsibility are doing more than their share. We also noticed some evidence, however slight, of friction between departments. This is not due to lack of cooperation, which is given when requested, but rather to lack of coordination."

"The salary scale of staff employees, excluding department heads, is in most cases in line with prevailing market levels. There are, however, several employees holding responsible jobs whose salaries are lower than they should be."

At a meeting of the AIME Executive and Finance Committees, November 14, authorization was voted for the employment of a business manager as proposed. Other recommendations are being studied.

Mining Branch Plans Varied Technical Program at Annual Meeting

PLANS are getting underway for the Annual Meeting of AIME, Feb. 18 to 21, 1952, in New York, with the various divisions of the Mining Branch lining up subjects for technical meetings. A brief resume of what to expect in technical sessions in the Mining Branch shows that emphasis is being placed on practical mining problems and increasing mine production.

The Mining Subdivision will hold seven sessions, of which two probably will be joint meetings, one with the Geology Subdivision on exploratory drilling and the other with the Industrial Minerals Div. on mechanized mining. Two sessions will include a symposium on drilling, covering such subjects as drill bits, drill steel, and drilling machines. A session on open pit mining practice will emphasize drilling and blasting, arranged by W. H. Goodrich.

The Canadian mining industry will sponsor a session describing several of their foremost operations, including block caving at the Helen Iron Mine, new developments by International Nickel at Sudbury, and Canada's largest gold mine—the Kerr-Addison.

The general session will include a special request report on *"Trends of Underground Mining in the Great Export Mines in Sweden,"* by Borje Hortzberg-Nordlund, chief engineer of the large iron ore producers of that country. Other interesting papers will deal with channel sampling with diamond-impregnated wheels and hydraulic hoisting.

The joint session on exploration and development drilling, developed by Robert Longyear, highlights prospecting by both the churn drill and the diamond drill, as well as the use of diamond drill data in the layout of a mining operation. The joint session on mechanical mining emphasizes new developments, particularly trackless equipment in both nonmetallic and metal mines. E. P. Pfleider is program chairman.

Geology Subdivision has planned four sessions. Three will be joint meetings, (1) with the Mining Subdivision on exploration and development drilling; (2) with the Geophysics Subdivision, and (3) with the Society of Economic Geologists. For the fourth session several general papers are available. Program committee chairman is G. M. Schwartz.

The Coal Div.'s nine sessions will extend from Monday through Thursday. Three sessions will be devoted to utilization and, as a part of these sessions, a number of foreign experts will review European practice in the gasification and liquifaction of coal. Two sessions will be devoted to mining; two sessions to preparation, one of which will be a symposium on heavy density equipment and procedures; one session to coal geology; and there will be a general session. Orville R. Lyons is chairman of the program committee.

The Minerals Beneficiation Div., plans still tentative. On Monday afternoon there will be symposium on *"What's New in Milling?"* Tuesday morning's two sessions are on pyrolysis and agglomeration; and jointly with the Industrial Minerals Div. with papers of mutual interest from both groups. Tuesday afternoon's two sessions will be on materials handling and automatic controls; and on manganese, to be held jointly with the Extractive Metallurgy and Iron and Steel Divisions. Sessions on Wednesday morning and afternoon cover crushing and grinding; Thursday morning, operating and automatic controls; and Thursday afternoon, flotation theory. Chairman, MBD program committee, is E. H. Crabtree, Jr.

Industrial Minerals Div. has in prospect one or two sessions on sulphur and sulphuric acid, two sessions on cement and lime, a symposium on ceramic raw materials, and a session on mining geology subjects. A joint session with the Minerals Beneficiation Div. will feature primarily beneficiation, flotation, mineral structure, etc., of titaniferous materials. One or two other joint sessions with other groups are indicated. Sanford S. Cole, vice-chairman, program committee, is arranging the technical fare for this division.

Geophysics Subdivision sessions will be devoted to papers on *"Aeromagnetics—Technical Aspects and Results," "Geochemistry-Geophysical Surveys," "Geophysics and Civil Engineering"* and *"Geophysics in Mining."* Leaders in the field of aeromagnetics are planning discussions relating to mining. Examples of the correlation between geochemistry and geophysical results are promised. The session on *"Geophysics in Mining"* will be worked into a joint session with the Geology Subdivision, with two papers: (1) Unique Methods in Mineral Exploration, "Rudore" and (2) Results of Radio Active Measurements from the Air.

The Banquet Committee announced that the banquet will be held in the Grand Ballroom of the Waldorf Astoria Hotel on Wednesday, February 20 at 7:30 pm.

The Waldorf Astoria is prepared to furnish private rooms for groups desiring to hold cocktail parties prior to dinner. Arrangements for these rooms should be made directly with the hotel.

The Committee will adhere firmly to the practice of assigning seats in the order of receipt of paid reservations. Tables will accommodate ten persons and those desiring to be seated with friends should purchase tickets in a block and indicate the name of the guest to whom the ticket will be issued. Reservation cards will be mailed to all AIME members early in January.

Publications' Policies And Prices Established for Coming Year

Pursuant to Article X of the bylaws of the AIME, the following information is hereby given as to the "conditions, prices, and terms under which the various classes of members, and Student Associates, severally, shall be privileged to obtain publications of the Institute during the ensuing year."

Publications authorized for 1952 publication include the following: MINING ENGINEERING, published monthly, containing material, including technical papers, of interest to those engaged in exploration, mining geology and geophysics, and metal, nonmetallic, and coal mining and beneficiation, and fuel technology. The JOURNAL OF METALS, published monthly, containing material, including technical papers, of interest to those engaged in nonferrous smelting and refining, iron and steel, and physical metallurgy. The JOURNAL OF PETROLEUM TECHNOLOGY, published monthly in Dallas, containing material, including technical papers, of interest to those engaged in petroleum and natural gas drilling and production.

Annual subscriptions to any one of the above journals will be provided all members in good standing without further charge. (A member ceases to be in good standing if current dues are not paid by April 1.) If more than one of the monthly journals is requested, $4 extra will be charged for an annual subscription, or 75¢ for single copies of regular issues and $1.50 for special issues. The nonmember subscription price for each journal is $8 in the Americas; foreign, $9. Student Associates will be entitled to the same privileges for all publications as members.

Three volumes of "*Transactions*" are authorized for 1952 publication, as follows: No. 190, Mining Branch; No. 191, Metals Branch; and No. 192, Petroleum Branch. These volumes will be available to members at $3.50 each for a first copy if paid for in advance with dues; otherwise at the nonmember rate of $7 less 30 pct. Nonmembers $7 in the United States; foreign $7.50.

Special volumes now planned for publication in 1952 include the following: 1—*Open Hearth Proceedings*, price to AIME members $7; nonmembers, $10. 2—*Blast Furnace, Coke Oven, and Raw Materials Proceedings*, AIME members $7; nonmembers $10. 3—*Electric Furnace Steel Proceedings*, AIME members $7; nonmembers $10. 4—*Symposium on Tube Producing Practice*, $3. 5—*Statistics of Oil and Gas Development and Production*, covering data for year 1950, free to members (except Student Associates); nonmembers $6. 6—*Statistics of Oil and Gas Development and Production*, covering data for year 1951, members $5, nonmembers $10.

If dues are paid subsequent to January 31, back issues of Institute publications will be supplied only if adequate stocks are on hand. A member may not receive a volume of "*Transactions*", or a special volume, in lieu of a monthly journal, free of charge on membership. Members in arrears for dues are not entitled to special members' prices for publications.

Rocky Mountain members may have their choice of an annual subscription to one of the monthly journals on request.

Institute Allows Military Credit

Pursuant to Article I, Section 5 of the bylaws as amended, giving the Board power to waive the age limitation for Junior Membership in the case of veterans from military service, the Executive and Finance Committees AIME voted to allow all veterans from military service who apply for Junior Membership a credit of one year on the entrance age limit of 30 years for each year of military service. Also, it was voted to allow a similar credit on the 33 year age limit for changing status to Associate Member or Member.

Fuels Conference Highlights Coal Preparation; Steam Plant Operation

Approximately 300 members, ladies and guests attended the 14th annual Fuels Conference, Coal Div. AIME-Fuels Section ASME, at Roanoke Hotel, Roanoke, Va., Oct. 11 and 12. There were four technical sessions, two of AIME origin and two by ASME. The former were devoted to a symposium on thickening and desliming and a foursome of papers on unrelated subjects. The ASME sessions were panel discussions.

C. E. Miller, chairman, ASME Fuels Div., presided at the luncheon on Thursday, at which R. H. Smith, president, Norfolk & Western Railway Co., was the speaker. C. A. Garner, chairman, Coal Div. AIME, presided at the banquet Thursday evening at which Fred K. Prosser was toastmaster.

The Percy Nicholls Award for 1951 was presented to Albert R. Mumford. The speaker was Walter S. Newman, president, Virginia Polytechnic Institute, who called on industry to help protect American colleges by enabling them to keep skilled staffs. Industry and Government have hired so many teachers away from the colleges that they find it difficult to do the job expected of them.

D. R. Mitchell and Orville R. Lyons presided at the first of the AIME sessions, Thursday morning. James P. Blair described typical installations of cyclone thickeners to clarify their role in the preparation of coal and to point out their versatility. He emphasized that each potential application requires a thorough study because of the variables involved. D. A. Dahlstrom discussed the desliming of fine solids, pointed out that a considerable amount of fine coal can be recovered as a salable product by sufficient elimination of high ash and moisture contributing slimes. "As desliming can be accomplished by simple classification at about the 200-mesh point, which will provide maximum recovery of coal, the liquid-solid cyclone appears as the likely tool." K. Prins described the application of the Prins *Streamcleaner* in the coal industry and emphasized the arrangement of the machine, its simplicity and trouble-free operation. H. R. Hagen, John Griffen, and Victor Phillips, reporting separately, presented operational data on the *locust summit thickener*.

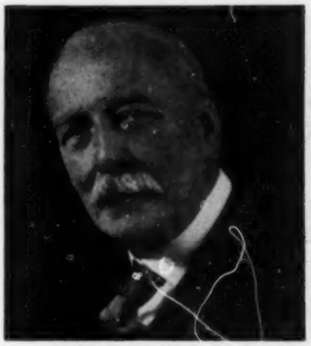
Two of the papers presented at the second AIME session Friday morning are in the October Transactions section of MINING ENGINEERING, namely, that by E. Swartzman and G. C. Behnke on the determination of free swelling index of coal employing electric furnaces and heaters, and the paper by Orville R. Lyons on filter cake size-consist and moisture relationships. Thomas F. Downing, Jr., reported on changing characteristics of storage coal, a study prompted by operating difficulties encountered while burning stored coal at Richmond Station in 1941. Henry F. Hebley discussed air pollution from gob piles, describing some of the control practices.

The ASME panel discussions, Thursday afternoon, were on engineering service in the coal industry. U. B. Yeager, George P. Cooper, Max A. Tuttle and E. J. Kerr contributed. Friday afternoon discussions were on fuel and equipment consulting service for small steam generating plants. At the latter, respective viewpoints were presented as follows: Of the small plant owner, by A. R. Miller; coal producer, by Earl C. Payne; consulting engineer, by H. C. Carroll; equipment manufacturer, by E. C. Webb (read by H. B. Moy), and construction, by W. S. Major.

The committee handling the conference consisted of: C. T. Holland, general chairman; F. K. Prosser, cochairman; O. R. Lyons, program AIME; E. C. Payne, program ASME; J. A. Hagy, Central Appalachian Section AIME; C. E. Pond, Virginia Section ASME.

Father and Son "Legion of Honor" Members of Institute

Andrew S. McCreath and Lesley McCreath, father and son, Harrisburg, Pa., together have been members of AIME for 100 years. Andrew S. McCreath joined in 1879 and was manager of the Institute in 1882. He died in 1930. Lesley McCreath joined AIME in 1902 — which adds up to the astounding total of 100 years.


ANDREW S. McCREATH


LESLEY McCREATH

The elder Mr. McCreath was born in Ayr, Scotland, in 1849, and came to United States in 1869. He was graduated from Andersonian University, Glasgow, and was doing postgraduate work at the University of Gottingen, when he was employed by the old Pennsylvania Steel Co., Steelton, Pa., as a chemist. He was one of the first men ever employed exclusively as a chemist by any steel company in United States. Subsequently he became chief chemist of the Pennsylvania Second Geological Survey, and when the survey was completed he established himself as an independent analytical and consulting chemist in Harrisburg.

Lesley McCreath, the son, was graduated from the Sheffield Scientific School of Yale University in 1901. He worked as chief chemist of Magnolia Sugar & Railroad Co., Lawrence, La. He subsequently worked in the coal fields of southeastern Kentucky correlating the various coal beds to show their continuity from Pennsylvania through West Virginia and Kentucky. In 1904, he joined his father's firm as a partner.

Andrew S. McCreath became a Legion of Honor member of AIME in 1929, and Lesley McCreath will become a member of the Legion of Honor in February, 1952.

Nominating Committee For 1953 Officers Named

As provided in Article IX, Section 2, of the AIME Bylaws, the names and addresses of the Nominating Committee for Institute officers for 1953 are given below. In each case, the name in parenthesis is that of the alternate, who will serve only in the absence of the principal at the meeting of the Committee.

Members designated by the Council of Section Delegates: Francis Cameron, St. Joseph Lead Co., 250 Park Ave., New York (Philip D. Wilson, Lehman Bros., 1 William St., New York); Henry A. Dierks, Glen Alden Coal Co., 631 Charles Ave., Kingston, Pa. (D. C. Helms, Lehigh Navigation & Coal Co., Lansford, Pa.); P. G. Spilsbury, Anaconda Copper Mining Co., Shoreham Hotel, Washington, D. C. (Thomas H. Miller, Bureau of Mines, Washington 25, D. C.); William A. Mueller, Ohio State University, Columbus (Harley Lee, Basic Refractories, Inc., 845 Hanna Bldg., Cleveland); Thomas M. Broderick, Calumet & Hecla Consolidated Copper Co., Calumet, Mich. (Raymond D. Satterly, Inland Steel Co., Box 360, Ishpeming, Mich.); Armor B. Martin, Montana Power Co., 40 E. Broadway, Butte (Tom Graham, Anaconda Copper Mining Co., Great Falls, Mont.); Drury A. Pifer, University of Washington, 505 Boston St., Seattle 9. (Albert H. Mellish, American Smelting & Refining Co., Tacoma, Wash.); George D. Dub, 1206 Pacific Mutual Bldg., Los Angeles 14, Calif. (alternate not selected); Edward M. Tittman, American Smelting & Refining Co., Box 1111, El Paso (Guy E. Ingersoll, Texas Western College, El Paso); S. S. Clarke, Tri-State Mines, Eagle-Picher Mining & Smelting Co., Cardin, Okla. (John W. Chandler, Eagle-Picher Mining & Smelting Co., First National Bank Bldg., Miami, Okla.).

Members designated by the Branch Councils: Edwin R. Price, Coal Properties, Inland Steel Co., Wheelwright, Ky. (J. Murray Riddell, dept. of mining engineering, Michigan College of Mining & Technology, Box 144, Houghton, Mich.); C. H. Behre, Jr., Geology Dept., Columbia Univ., New York 27 (Grover Holt, Cleveland-Cliffs Iron Co., Hibbing, Minn.); John D. Sullivan, Battelle Memorial Institute, 505 King Ave., Columbus 1 (Maxwell Gensamer, dept. of metallurgy, Columbia University, New York); John E. Sherborne, Union Oil Co. of California, 1004 Summit Drive, Whittier, Calif. (Lloyd Elkins, Stanolind Oil & Gas Co., P.O. Box 591, Tulsa 2).

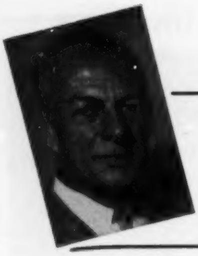
Members designated by President Peirce: John R. Suman, Chairman, Standard Oil Co. (N. J.), 30 Rockefeller Plaza, New York 20. (Henry T. Mudd, 1206 Pacific Mutual Bldg., Los Angeles, Calif.); Kent R. Van Horn, Aluminum Co. of America, 2210 Harvard Ave., Cleveland (George P. Halliwell, H. Kramer & Co., 1339-45 W. 21 St., Chicago 8); H. M. Griffith, The Steel Co. of Canada Ltd., Hamilton, Ont., Canada (C. D. King, U. S. Steel Corp., 436 7th Ave., Pittsburgh).

The Committee will welcome suggestions for the official slate for 1953. The following offices are to be filled: President-Elect for 1953 (President for 1954), two Vice-Presidents, and six other Directors. The terms of the following Directors eligible for reelection expire in February 1953: D. H. McLaughlin, R. W. Thomas, H. Decker, F. B. Foley, E. C. Meagher, C. V. Millikan, G. F. Moulton, and Howard I. Young.

Legion of Honor

Class of 1902

Addicks, Lawrence, Bel Air, Md.; Barker, H. A., Santa Fe, N. M.; Boyd, W. S., San Francisco, Calif.; Bradford, Seymour K., No address; Brown, Davenport, Boston, Mass.; Cornell, Russell T., Scarsdale, N. Y.; Crawford, H. E., Long Beach, Calif.; Emery, Augustus Bachelder, Bryanston, Transvaal, So. Africa; Eustis, Augustus H., Boston, Mass.; Eustis, Fredric A., Boston, Mass.; Fleming, Edward P., Los Angeles, Calif.; Fuchs, Fernando C., Lima, Peru; Jameson, A. H., Branford, Conn.; Landfield, J. B., San Francisco, Calif.; Leggat, Alexander, Butte, Mont.; Lidgerwood, John H., Beacon, N. Y.; McCreath, Lesley, Harrisburg, Pa.; Miller, D. I., Coshocton, Ohio; Morris, Henry C., Washington, D. C.; Oliver, Edwin Letts, San Francisco, Calif.; Pringle, Robert W., Salisbury, S. Rhodesia, Africa; Queneau, A. L. J., New York City; Roberts, Milnor, Seattle, Wash.; Russell, William, Reigate, Surrey, England; Sales, Reno H., Butte, Mont.; Smith, Hoval Arnold, Phoenix, Ariz.; Smith, Howard D., New York City; Sproat, A. D., Guanajuato, Gto., Mexico; Utley, H. H., Baxter Springs, Kans.; Walker, Etheredge, San Francisco, Calif.; Whitney, W. R., Schenectady, N. Y.; Wickes, L. Webster, Los Angeles, Calif.



THE DRIFT OF THINGS

by Edward H. Robie

In this space this month we had expected to give our impressions of Mexico. Our plan of driving to the Fall Meeting in Mexico City, however, was unfortunately interrupted by an attack of bronchitis to which both the distaff and the mistaff sides of the family succumbed. As a result, several days were spent in Austin, Texas, instead of in Mexico City. We hear the meeting was a great success, but that is only hearsay.

We did fly to Los Angeles for an interesting and well attended Board meeting on October 25. It was a happy thought of Fay Libbey to take advantage of the Los Angeles meeting of the Mining Congress, the Pacific Petroleum Chapter of AIME, and that of our Southern California Section to schedule an extra Board meeting. Board meetings should be dispersed around the country as much as practical.

Also, we spent three interesting hours at the University of Texas, taking advantage of Harry Powers' hospitality; enjoyed a dinner meeting until nearly 11 pm with some of the wheel horses (if there can be more than one wheel horse) of the Gulf Coast Section; and attended a similar luncheon meeting with 12 of the Delta Section in New Orleans. Professionally, we know less about what the petroleum boys do than we do about the activities of the mining and metals groups, but still we get a particular kick out of talking to oil men. In general, they are a younger group and have more ideas for changes in the Institute—things that we are paid to think of ourselves. Also, they are exceptionally friendly and hospitable. Further, they are critical of our mistakes, so keep us on our toes.

In New Orleans, we were especially indebted to Messrs. Petree and Hatfield of the Gulf Oil Co. for a thrilling airplane ride over the oil fields in the Delta Country of the Mississippi. Ol' Man Ribber has several mouths, but a mere look at a map gives but a poor idea of them. Water—in ponds, channels or marshes—covers almost the entire area, but under the water, even out to sea, are vast deposits of gas, oil, and sulphur. Some of the gas is burning for lack of pipe lines to carry it to market, but the occasional flares are not too conspicuous. Tracks of the marsh buggies, man-made amphibians that travel over the marshes and swim when they come to open water, are a feature of much of the area as seen from the air. Oil wells in the water have no derricks except those wells being drilled, but look like enlarged buoys of some kind. At one point, we frightened into flight thousands of ducks which had sense enough to arrive well ahead of the mid-western storms of early November.

No further offshore exploration is being done pending final determination of whether the Federal Government or the States are to have the right to lease the lands. We gather that it doesn't make too much difference to the oil companies, but they would like to have the question finally decided.

Our trip will culminate in a couple of days in the phosphate country with a visit to our Florida Section at Lakeland. Then home.

Oil has long been a cause of international friction and the situation in Iran might easily have started a new war if cool heads had not prevailed. On paper and following the old rules, the British had a right to own and operate their Iranian oil properties. Times are different now, however, and American oil companies, at least, have learned that exploitation of native populations leads to trouble. Sharing profits and assistance in the development of a country have proved a better policy, regardless of rights that might have been given in the past. Even so, Communistic influences are always ready to urge complete nationalization of any conspicuously successful foreign enterprise. Operations in a foreign country are always a gamble. In fact, with the United States itself about one-third socialized, capital is not safe anywhere any more. We say one-third because this is said to be approximately the portion of our national income which we perforce must pay to our Government.

Getting back to Iran and the Abadan refinery, it seems that this is exactly the type of problem that the United Nations should be set up to resolve. It is certainly a problem that concerns a foreign government or foreign nationals, as well as Iran. International law should state plainly what the respective rights are in a situation such as this, and the United Nations should have the power to enforce these rights. Only by such application of the idea of World Government can the United Nations develop into much more than a debating society.

The year-end financial reports are not available yet, of course, but we can say with some assurance that the Institute will finish 1951 with a sizable balance in the black. Not since 1941 has this occurred without transferring money from one or more of the Funds. In this period, some $220,000 has thus been added to income to meet deficits. Practically no more money is available from such funds, so it is fortunate that none will be needed.

The money that has been used in founding three new magazines and building them up to a high standard is showing results. Increased advertising income is largely responsible for the improved income. Unit expenses have increased with the continued decline in the value of the dollar, but rigid economy and the absence of a Directory in 1951 should hold expenses to about the 1950 figure.

Some of the 1951 profit is badly needed for new equipment at AIME headquarters and for an additional staff member. The rest should be used in making a start on repayment of the money borrowed from the reserve funds.

What of 1952? Barring a still further important drop in the value of the dollar—which is a dangerous assumption—the income next year should exceed considerably that of this year and expenses should not rise by as much. So, the net should be even better next year.

What of 1953? That depends upon the Referendum on dues, which will be decided by the members in the spring or summer of next year. We feel now that the future of the Institute would be adversely affected to a serious degree if the dues in 1953 went back to the 1922-1949 level. We also feel that by a further investment in the magazines and rehabilitation of the Funds, there is an excellent possibility that much greater services can be extended to members five or ten years hence than at present; or that the present services can be given for less dues income. But that statement is made only if the value of the dollar does not decline further. This, we confess, is a big "if."

Personals

O. Perry Riker is with the ECA at Bangkok, Thailand, lecturing at Chulalongkorn University.

Marston Fleming is in the United States on a Nuffield Traveling Fellowship visiting plants, research laboratories, and universities. He is senior lecturer at the Royal School of Mines in England.

Raymond E. Zimmerman is now chief preparation engineer, coal div., U. S. Steel Co., Pittsburgh. He had been with the Koppers Co., Inc. at Zonguldak, Turkey.

Norman C. Prudent has become mine superintendent for the Southwest Potash Corp., Carlsbad, N. Mex.

Herman F. Brownbill is general manager for the Cia. American Smelting Boliviana S.A., La Paz, Bolivia.

Ralph W. Marsden is now with the Oliver Iron Mining Co., Duluth, Minn.

Clyde N. Garman has left Catavi, Bolivia and is returning to the United States.

Donald G. Finlayson expects to return to Montreal from Labrador shortly. He is with the Iron Ore Co. of Canada.

Raymond C. Troxell is doing graduate work in the mineral preparation div. at Penn State College. He has been appointed a graduate assistant and is an instructor for several undergraduate laboratories.

Fred Lee Andersen is now district sales manager for the Thew Shovel Co., Aurora, Colo.

A. H. Kapadia has enrolled for graduate study at the University of Minnesota.

Sylvester A. Hanson is now with the Reserve Mining Co., Babbitt, Minn.

Charles J. Hager, engineer, Stith Coal Co., America, Ala., has resigned to accept the position of chief mining engineer, Alabama By-Products Corp., Birmingham.

L. Sonneveld has resigned as junior mining engineer with Mauricio Hochschild, S.A.M.I., Bolivia. He has joined the Sigma Mines (Quebec) Ltd., Bourlamaque, Que.

Gunther E. Rochefort has accepted a position with the Los Maquis mine, Cabildo, Chile.

Charles J. Short is with Kaiser Engineers, Inc., Jamaica, B.W.I. as assistant construction superintendent working on the bauxite project.

John E. Bowenkamp, Freeport Sulphur Co., is now in the New York office.

Channing B. Mould has joined Mining & Quarrying Associates, Keeseville, N. Y.

C. W. Hawn has resigned as mine superintendent with Cia. Minera Aguilar, S.A., El Aguilar, Argentina and is returning to the United States.

ROY H. GLOVER

Roy H. Glover has been elected a member of the board of directors, vice-president, and general counsel of the Anaconda Copper Mining Co. Mr. Glover has been serving as western general counsel of the company.

W. E. Sands is now metallurgist for the Black Rock Mining Corp., Bishop, Calif.

David E. Morgan is on an extended trip to Europe, Israel, Tunis, and Morocco. He is visiting mines, acting as mining consultant and foreign representative for major American engineering and mining equipment manufacturing companies. Mr. Morgan is vice-president of the International Mfg. & Equipment Co., New York and president of Peerless Precision Products Co., Providence, R. I.

Lloyd R. Jackson has been named an assistant director of Battelle Memorial Institute, Columbus.

Herbert Hoover has been awarded the Howard Coonley Medal. The medal is presented each year to an executive who has given outstanding service in advancing the national economy through voluntary standards.

Forbes Wilson, formerly with the Rising & Nelson Slate Co., Inc., West Paulet, Vt., is now with Freeport Sulphur Co., New York.

James Norman, formerly of the Bureau of Mines and Tennessee Coal, Iron & Railroad Co., has joined Senior, Juengling & Knall as head of the mining and metallurgical div., Birmingham.

Ben Messer is now mine engineer for Duval Sulphur & Potash Co. of Carlsbad, N. Mex. He was formerly with Miami Copper Co., Miami, Ariz.

Helen A. Antonova is with Robert Wesley Briggs, consulting engineer, working on design of the New England thruway in New Rochelle, N. Y.

Edwin Sweetman, mining engineer of Green Bay, Wis., recently returned from a visit to eastern Canada and United States.

Robert Youtz has resigned as shift boss, Resurrection Mining Co., Leadville, Colo. to accept the position of quarry engineer with U. S. Gypsum Co., Southard, Okla.

T. G. Baker is presently with DMA as chief of the Foreign Div., Washington, D. C. Mr. Baker is a consultant for Lake George Mines, Ltd.

Dave E. Wick, formerly with Kaiser Steel Corp., is now a mining engineer with Reserve Mining Co., Babbitt, Minn.

Keith Whiting, formerly chief geologist for the northwest div., American Smelting & Refining Co., Wallace, Idaho, has been made chief exploration engineer, Salt Lake City. He succeeds **Lyman M. Hart,** who has been transferred to the New York office. **M. W. Cox** will succeed Mr. Whiting at Wallace.

Justin B. Gowen, consulting engineer for Manganese Products, Inc., Seattle, is now with the Defense Minerals Production Authority, Washington, D. C.

Harry L. Miller is now general superintendent, American Zinc, Lead & Smelting Co., Ouray, Colo. He had been assistant mine foreman with the N. J. Zinc Co., Gilman, Colo.

Richard F. Brooks has become production manager for the U. S. Pumice Supply Co., Inc., Leevinning, Calif.

Edgar C. Long, Baroid Sales Div., National Lead Co., Los Angeles has been transferred to Houston.

Robert G. Woods has left the employ of the Yucca Mining & Milling Co., Yucca, Ariz. and has joined the Rhude Media Co., Marble, Minn., as plant superintendent.

Robert Y. Grant has resigned as chief, mining and geology div., Natural Resources Section, General Headquarters, SCAP.

John D. Ridge was appointed chief of the div. of mineral economics, Pennsylvania State College, State College, Pa.

PAUL G. BASTIEN

Paul G. Bastien, Schneider & Co., Paris, has been elected president of the International Committee on Testing Cast Iron, Brussels.

Joseph Conrad Twinem is now chief of the Miscellaneous Metals & Mineral Section, Metal Branch, Industrial Materials and Manufactured Goods Div., Office of Price Stabilization, Washington, D. C.

Robert B. Davis is now employed as a mining engineer with the Cleveland-Cliffs Iron Co., Ishpeming, Mich.

A. George Komadina has been made assistant mill superintendent, central mill, Eagle-Picher Mining & Smelting Co., Miami, Okla.

Robert J. Piros, Jr. has joined the International Minerals & Chemical Corp., Florida experiment station at Mulberry, Fla. He had been with the Truax-Traer Coal Co., Pinckneyville, Ill.

Edwin Riley Servis is now with the Sharey Northern Peru Smelting & Mining Co., Trujillo, Peru.

Louis P. Starck has resigned as superintendent of the tungsten concentrator, Canadian Exploration Ltd., Slamo, B. C.

Lester R. Brown, Jr. is now mine superintendent of the Mina El Dorado, San Isidro, El Salvador. He had been with the Compagnie Aramayo de Mines in Bolivia.

William Wentworth has taken a position with the American Smelting & Refining Co., Trujillo, Peru as mill superintendent.

John Kenneth Jones is with the International Smelting & Refining Co., Salt Lake City.

Kenneth Frederick Packer has become an instructor at the University of Michigan, Ann Arbor.

ROBERT HENDERSON

Robert Henderson and **Charles A. Cleeves** have joined the mining div. of E. J. Longyear Co., Minneapolis. Mr. Henderson had been a consultant on mining problems. Mr. Cleeves has been engaged in operating, engineering and mining for several years.

Obituaries

Robert Moffitt Black (Member 1909), professor at the University of Pittsburgh, School of Mines, died on Sept. 10. Dr. Black was born in Meyersdale, Pa. and attended Vanderbilt, Harvard, Michigan College of Mines, and Wisconsin University. After spending a short time as a practicing engineer, he was an instructor in civil engineering at the State University of Iowa. In 1912 he joined the University of Pittsburgh as assistant professor of mining.

Walter Lyman Brown (Member 1923) died on Sept. 5 at Carmel, Calif. Born in 1880 at San Francisco he attended the University of California and graduated in 1903 with the degree of B.S. in mining. His practical experience was gained in Mexico, Alaska, California, and at Bunker Hill & Sullivan, Idaho. Three years were spent in South Africa at the New Modderfontein and Crown mines. He then went to the Gold Coast, West Africa as assistant manager of the Abboutiakoon mine. During the first world war he joined the Commission for Relief in Belgium as director. After the war he continued to work on the relief and rehabilitation of Europe. In 1923 he was in Moscow negotiating concession agreements for the Lena Goldfields Co. He returned to California in 1934 to become president of the Carson Hill Gold Mining Corp. For his relief work he was decorated by the French, Belgian, Austrian and Polish governments.

Memorial Resolution

Holcombe J. Brown

The following resolution was prepared by George P. Swift on the death of **Holcombe J. Brown** a former Director of the Institute:

Whereas, Almighty God, Supervisor of all, has taken His right to remove from his family, friends and associates Holcombe J. Brown whose long affiliation and work with the American Institute of Mining & Metallurgical Engineers was recognized by two terms as Director and one term as Vice-President and Director and

Whereas, his wide technical knowledge and practical experience were used to advantage by the Engineering Societies of New England as their President and Councillor for many years and

Whereas, his untiring efforts contributed considerably to the success and operation of the Massachusetts State Board of Professional Engineers and Land Surveyors since its inception, now therefore be it

Resolved by the Board of Directors of the American Institute of Mining

& Metallurgical Engineers, at their meeting held Wednesday, Sept. 12, 1951, that we stand in silent prayer in memory of our departed colleague with whom we have worked long and hard, and be it further

Resolved that this resolution be placed on the records of this Society and a copy, suitably engrossed, be sent to the family of the late Holcombe J. Brown.

A. F. Connell (Member 1948) died on Jan. 1. He attended Bethlehem Prep. School and Lehigh University. During the first world war he served overseas and in 1919 became manager, East Alden Mining Co. and then joined Repplier Coal Co., Buck Run, Pa. as vice-president, manager.

Leonidas C. Glenn (Member 1900) died on January 11. Born in 1871 at Crowdus Creek, N. C. he attended the University of South Carolina, Harvard, and Johns Hopkins University. After several years of field work in geology he joined the U. S. Geological Survey and then was professor of geology at Vanderbilt University, Nashville, Tenn.

Claude Garrison Grim (Member 1940) died Sept. 8, 1951, at Boron, Calif. He received a B.S. in Mining Engineering in 1926 from Iowa State College, and was actively engaged in mining at the time of his death. He had worked for Phelps Dodge Co., Morenci, Ariz. from 1924 to 1932 and was underground superintendent of transportation when he left the company to become engineer in charge of production and research for U. S. Potash Co., Carlsbad, N. Mex. In 1937, he was made division foreman of Climax Molybdenum Co., at Climax, Colo., and in 1939 he joined Pacific Coast Borax Co., Boron, Calif., as assistant mine superintendent. He was with this company at the time of his death.

Garnett Alfred Joslin (Member 1917) died Sept. 20, 1951 in Mexico City at the age of 68. Born in Fargo, N. D., he graduated in 1909 with a B.S. in mining from Massachusetts Institute of Technology. His early work was on examination and management of mine in Cobalt and Porcupine, Ont., and was later superintendent of the Johnnie Mine, Goldfield, Nev.; geologist for Caribbean Petroleum Co., Venezuela; and later in examination and operating work in western United States, Mexico, and Chile. In 1927 he established a consulting firm in Los Angeles, and he was an associate member of Behre Dolbear & Co., New York. In 1947 he moved to Mexico where he served as consultant to Cia Metallurgica Mexicana until his death.

Albert Edward Marshall (Member 1916) of Providence, R. I., died Sept. 15, 1951. He was born and educated in Liverpool, England, and at the time of his death had a chemical consulting organization. From 1902 to 1908, he was employed by United Alkali Co., England, connected mainly with mineral acids. From 1908 to 1912 he was employed as a chemical engineer with Thermal Syndicate, England, and in 1912 came to New York as engineer and manager for the same company. In 1916, he became works manager for Davison Chemical Co., Baltimore, and in 1921 he became a consulting chemical engineer.

Thomas Murphy (Member 1928) has died. Mr. Murphy was born in Wyoming County, W. Va. on Aug. 16, 1886. From 1901 to 1903 he was a helper on mechanical and electrical work in mines for the Algoma Coal & Coke Co., Algoma, W. Va. He then joined the Pulaski Iron Co., Eckman, W. Va. and in 1905 was a motorman for the Zenith Coal & Coke Co., Crumpler, W. Va. Mr. Murphy went to Montana and joined the North Western Improvement Co. at Red Lodge. He was motorman, chief electrician, and was promoted to master mechanic and chief electrician. In 1925 he was transferred to Roslyn, Wash. as superintendent of Roslyn Field.

A. M. Strong (Member 1919) died on July 14, 1951. He received his A.B. in chemistry from Stanford University in 1899. He spent some time prospecting and in 1901 became assayer for the Reward Gold Mining Co. He

opened his mining and civil engineering practice at Independence, Calif. During this time he had charge of special work for the Darwin Mining Co., Four Metals Mining Co., Black Canyon Mining Co., Bishop Creek Gold Co. and others. He later moved to Los Angeles. Mr. Strong had been elected a Senior Member of AIME just before his death.

NECROLOGY

Date Elected	Name	Date of Death
1936	Charles N. Becker	May 1951
1909	Henry T. Beckwith	Oct. 5, 1951
1921	Adolph Bregman	Oct. 4, 1951
1937	E. L. Clair	Unknown
1903	Will L. Clark	Sept. 4, 1951
1937	Robert S. Cockle	Sept. 21, 1951
1948	Andre Delruelle	July 23, 1951
1950	John Eck	Sept. 24, 1951
1922	Edward Griffith	Oct. 24, 1951
1944	J. H. Kerrick	October 1951
1947	Harry A. Knowlton	Aug. 31, 1951
1951	C. A. Norbury, Jr.	Unknown
1938	Hugh J. MacLean	Sept. 19, 1951
1937	Robert D. Maddox	Unknown
1903	Elwyn W. Stebbins	May 21, 1950
1911	Paul Sterling	Aug. 14, 1951
1922	John J. Tweedie	Unknown
1903	H. D. Williamson	Oct. 14, 1951

Proposed for Membership

MINING BRANCH, AIME

Total AIME membership on Oct. 31, 1951, was 17,313; in addition 2603 Student Associates were enrolled.

ADMISSIONS COMMITTEE

Thomas G. Moore, Chairman; Carroll A. Garner, Vice-Chairman; George B. Corless, F. W. Hanson, Albert J. Phillips, Lloyd C. Gibson, R. D. Mollison, John T. Sherman. Alternates: A. C. Brinker, H. W. Hitzrot, Plato Malozemoff, Ivan Given, T. D. Jones, and W. A. Clark, Jr.

Institute members are urged to review this list as soon as the issue is received and immediately write the Secretary's Office, night message collect, if objection is offered to the admission of any applicant. Details of the objection should follow by air mail. The Institute desires to extend its privileges to every person to whom it can be of service but does not desire to admit persons unless they are qualified. Objections must be received before the 30th of the month on Metals and Mining Branches.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

Arizona
Globe—Weaver, Myron B. (A)
Superior—Davis, Franklin T. (M) (C/S—J-M)
Superior—Wallach, Albert A. (M) (R. C/S—S-M)

Arkansas
Bauxite—McBride, George C. (M) (C/S—J-M)

California
Altadena—Pray, Lloyd C. (M) (C/S—J-M)
Claremont—Ball, Emmett B., Jr. (J) (C/S—S-J)
Claremont—McKenna, Donald C. (M)
Sacramento—Peterson, Donald W. (J) (C/S—S-J)
San Mateo—Boman, Johan (Jan) E. (M)

Colorado
Denver—Storm, Howard A. (M) (R. C/S—J-M)
Grand Junction—Watts, James H. (A)

Idaho
Smelterville—Tapper, Thomas M. (C/S—J-M)

Illinois
Glen Ellyn—Feigin, Harry M. (M)

Kansas
Lawrence—Kulstad, Robert Otto (J)

Massachusetts
Cambridge—Koch, George S., Jr. (J)

Michigan
Ann Arbor—Heinrich, Eberhardt W. (M) (C/S—J-M)
Ferndale—Haworth, Roy D. (M) (C/S—J-M)
Houghton—Rand, John R. (J) (C/S—S-J)
Ishpeming—Stuart, Wilbur T. (M)

Minnesota
Hibbing—Kerr, Charles D. (M)
Hibbing—Merklin, Kenneth E. (M) (R. C/S—J-M)
Hibbing—Remer, Charles H. (M)
Hibbing—Webb, William D. (M)
Leoneth—Martin, James S. (M) (C/S—J-M)
Taconite—Hollett, Allen B. (J) (R.)

Missouri
Flat River—Benner, Denny R. (J) (C/S—S-J)

Montana
Butte—Ary, T. S. (J) (C/S—S-J)
Butte—Briney, Frank E. (J)
Butte—Drechsler, Herbert D. (J) (C/S—S-J)

Nevada
Las Vegas—Wells, Joe W. (A)

New Jersey
Bound Brook—Anderson, Edwin E. (J) (C/S—S-J)
Westfield—Wiendl, Joseph A. (M) (C/S—J-M)

New Mexico
Albuquerque—St. Germain, Arthur A. (J) (R. C/S—S-J)
Carlsbad—Atwood, George E. (M) (C/S—J-M)
Carlsbad—Christo, Sam, Jr. (J) (R. C/S—S-J)
Carlsbad—Foy, Harold E. (J)
Carlsbad—Keller, Frank Colvin, Jr. (A)
Carlsbad—Montgomery, Vol H. (M)
Grants—Rapaport, Irving (A)
San Lorenzo—Leake, Daniel B. (M)
Silver City—Thompson, Rodric R. (A)
Silver City—White, Wade A. (M)

New York
Mineville—Farrell, Patrick F. (M) (C/S—J-M)
New York—Leytess, Eugene (R. M)
Tahawus—Holland, John S. (M) (C/S—J-M)

Proposed For Membership

Ohio
Portsmouth—Kirk, Earl C. (M)

Pennsylvania
Cornwall—Neil, Charles R. (M) (C/S—J-M)
Irwin—Jones, Wilmot C., Jr. (J) (R. C/S—S-J)
Johnstown—Chedsey, George L. (M) (C/S—J-M)
Pittsburgh—McKee, John P. (M)

South Dakota
Rapid City—Cope, Joseph H. (A)
Rapid City—Stevens, Edward H. (M)

Tennessee
Louisville—Hill, William T. (J)
Mascot—Bumgarner, James G., Jr. (J) (C/S—S-J)

Texas
El Paso—Raffo, Frank MacLaughlin (M)

Utah
Bingham Canyon—Crandall, Lucian D. (J) (C/S—S-J)
Lark—Weagel, Robert C. (M) (C/S—J-M)
Salt Lake City—Frost, John E. (J) (C/S—S-J)
Salt Lake City—Siegel, Horace U. (A)
Salt Lake City—Wadsworth, Milton E. (J)
South Midvale—Helm, John E. (J) (C/S—S-J)
West Jordan—Chiea, Marsel (M)

Washington
Olympia—Huntting, Marshall T. (M) (C/S—J-M)
Spokane—Huckaba, J. Stanley (M)

West Virginia
Lansing—Naeve, Ennis Albert (M)
Morgantown—Wiebe, Donald (A)
Sharples—Price, Vance (J) (R. C/S—S-J)

Wyoming
Laramie—Wilson, William H. (A) (C/S—S-A)

Africa
Bogoso, Gold Coast—Goudarzi, Gus H. (M) (C/S—J-M)
Nigeria—Stevenson, William G. (M) (C/S—A-M)

Alaska
Fairbanks—Chapman, Robert M. (M) (C/S—J-M)

Australia
Newcastle—Whitehead, Robert C. (J) (C/S—S-J)

Bolivia
Leallagua—Van Nordheim, Rudolph (M)
Siglo—Zubrzycki, Piotr P. (A)

Canada
Alberta—Cochrane, Thomas Stirton (J) (C/S—S-J)
Quebec—Cowdery, Peter H. (J)

Chile
Rancagua—Garcia, Manuel R. (M)
Santiago—Cuevas, Francisco (M)

India
Calcutta—Sinha, Bindeshwari N. (M) (C/S—A-M)
Nagpur—Cassad, Dhunjisha P. R. (M)

Mexico
Altillo—Beecroft, Carl J. (M)
Durango—Myers, Henry Ernest (M)
Mexico City—Evans, Thomas A. (M)
Mexico, D. F.—Booth, Daniel A. (A)
Mexico, D.F.—Bolton, Henry E. (M) (R. C/S—J-M)
Mexico, D. F.—Griffiths, Gordon A. (J)
Taxco—Foster, Earl F. (M) (R. C/S—J-M)
Taxco—Gandara, Jesus Jose G. ( )
Taxco—Johnson, John C. (J)
Taxco—Newell, Harry A. (A)
Taxco—Stern, Roger (M)

Norway
Bjornevatn—Smith, Meyer (J)

Peru
Casapalca—Howard Goldsmith, Raymond C. (M) (C/S—J-M)
Lima—Ferreyros, Ernesto (A) (C/S—J-A)
Trujillo—Cooke, Herman R., Jr. (M) (C/S—A-M)

Spain
Madrid—Kabana, Manuel C. (J)

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Coming Events

Dec. 2-3, American Institute of Chemical Engineers, annual meeting, Chalfonte-Haddon Hall, Atlantic City, N. J.

Dec. 5, AIME, Chicago Section, Chicago Bar Assn., 29 S. La Salle St., Chicago.

Dec. 6-8, AIME, Electric Furnace Steel Conference, William Penn Hotel, Pittsburgh.

Dec. 7, AIME, Lehigh Valley Section, annual meeting, Hotel Bethlehem, Bethlehem, Pa.

Dec. 10, AIME, Arizona Section, annual meeting, Pioneer Hotel, Tucson.

Dec. 11, AIME, Morenci Sub-section, Longfellow Inn, Morenci, Ariz.

Dec. 26-31, American Assn. for Advancement of Science, annual meeting, Philadelphia.

Dec. 27-28, American Chemical Society, Div. of Industrial & Engineering Chemistry, symposium on nucleation, Northwestern University, Evanston, (Chicago).

Jan. 8, 1952, Society for Applied Spectroscopy, 6 pm, supper, Tosca's; 8 pm, meeting, Socony-Vacuum Training Center, 63 Park Row, New York.

Jan. 9, AIME, Chicago Section, Chicago Bar Assn., 29 S. La Salle St., Chicago.

Jan. 13-15, Institute of Scrap Iron & Steel, Waldorf-Astoria, New York.

Jan. 16-18, Society of Plastic Engineers, Inc., annual national technical conference, Edgewater Beach Hotel, Chicago.

Feb. 5, Society for Applied Spectroscopy, 6 pm, supper, Tosca's; 8 pm, meeting, Socony-Vacuum Training Center, 63 Park Row, New York.

Feb. 6, AIME, Chicago Section, Chicago Bar Assn., 29 S. La Salle St., Chicago.

Feb. 18-21, AIME, annual meeting, Hotel Statler, New York.

Mar. 3-7, ASTM, spring meeting and committee week, Hotel Statler, Cleveland.

Mar. 5, AIME, Chicago Section, Ladies' Night, Chicago.

Mar. 11, Society for Applied Spectroscopy, 6 pm, supper, Tosca's; 8 pm, meeting, Socony-Vacuum Training Center, 63 Park Row, New York.

Mar. 16-19, American Institute of Chemical Engineers, Atlanta Biltmore Hotel, Atlanta.

Mar. 22-Apr. 6, Chicago International Trade Fair, Navy Pier, Chicago.

Apr. 9, AIME, Chicago Section, Chicago Bar Assn., 29 S. La Salle St., Chicago.

Apr. 25-26, AIME, New England Regional Conference, Kenmore Hotel, Boston.

May 1-7, International Foundry Congress, Convention Hall, Atlantic City, N. J.

May 6-9, Scientific Apparatus Makers Assn., annual meeting, Edgewater Beach Hotel, Chicago.

May 11-14, American Institute of Chemical Engineers, Atlanta Biltmore Hotel, Atlanta.

May 22-24, American Society for Quality Control, annual convention, Onondaga County War Memorial, Syracuse, N. Y.

June 23-27, ASTM, 50th anniversary meeting, Hotel Statler, New York.

July 1-Sept. 30, Centennial of Engineering, Chicago.

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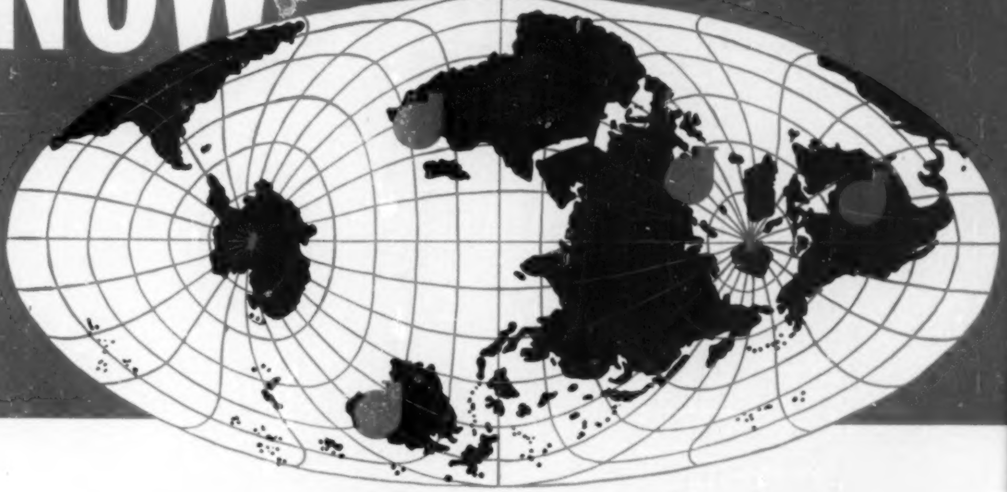

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HYDROSEAL, PACKLESS AND MAXIMIX DESIGNS ARE COVERED BY PATENTS AND APPLICATIONS IN THE MAJOR MINING CENTERS OF THE WORLD